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Vol. 109

METAL MINING AND NONMETALLIC MINERALS

1934

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PREFACE

Material in this volume, designated "Metal Mining and Nonmetallic Minerals," includes certain groups of technical papers that, with the matter that now goes into the YEAR BOOK, formerly constituted the so-called General Volume of Transactions. The groups include Metal-mining Methods, Ground Movement and Subsidence, Health and Safety in Mines, Nonmetallic Minerals, and Mineral Industry Education. None of these were represented in the TRANSACTIONS of 1933. It is believed that most members who are primarily interested in any one of these subjects are likely to have at least a secondary major interest in the others. The object is to attain the most effective distribution of papers from the standpoint of the entire membership with a minimum of waste in publication. The complete list of TRANSACTIONS volumes to be published this year is as follows:

PETROLEUM DEVELOPMENT AND TECHNOLOGY
COAL DIVISION
METAL MINING AND NONMETALLIC MINERALS
GEOPHYSICAL PROSPECTING
INSTITUTE OF METALS DIVISION
IRON AND STEEL DIVISION
MILLING AND ORE DRESSING (SPECIAL ROCKY MOUNTAIN VOLUME)

The only major subjects not included in these volumes are Nonferrous Metallurgy and Mining Geology. It will be recalled, however, that there appeared late in 1933 the special Rocky Mountain TRANSACTIONS volume on "Copper Metallurgy" and the Rocky Mountain Fund Series volume, "Ore Deposits of the Western States" (Lindgren volume). Because of these two exceptionally fine volumes, members most concerned with geology and "production" metallurgy will not feel slighted.

Each member is entitled to receive one of the volumes in the foregoing list of TRANSACTIONS without cost, and such others as he desires at the member's rate of $2.50 per volume. Each, of course, will receive his copy of the YEAR BOOK published at the end of the current year.

Credit should go to Messrs. Lucien Eaton, George S. Rice, John T. Ryan and S. H. Dolbear, chairmen during the year 1933–34 of the Committees on Mining Methods, Ground Movement and Subsidence, Health and Safety in Mines, and Nonmetallic Minerals, respectively, for their work in organizing the programs at which most of the papers were presented and for assembling and arranging the material in the volume. Likewise appreciation is due to the members of the Papers and Publications Committee and to the authors who contributed the excellent papers that compose the volume.

A. B. Parsons, *Secretary.*

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[1] Until February, 1935. [2] Until February, 1936. [3] Until February, 1937.

Selection of a Mining System

By Robert K. Warner,* New Haven, Conn.

(New York Meeting, February, 1933)

When a new mine is opened, and often when an operating mine must adapt itself to physical or economic changes, a mining system must be selected in complete detail. In the past the plan chosen was usually the product of the local staff alone, however limited the training and experience of that staff might be, which often resulted in the introduction of methods that were familiar to the staff, but not so well suited to the conditions as others unknown to the local men. A new management, noting the difficulties with the existing layout, might make modifications or complete changes, so that many mines in their history have seen a whole series of mining systems, each somewhat laboriously grafted on to its predecessor.

This should not be interpreted as a condemnation of the older staffs for sticking to the tried systems, such as open square-sets in the early days at the Homestake, because they knew it would work after a fashion, nor does the statement apply to the technically trained, and usually broadly experienced staffs of most present-day properties. The older miners did not have the variety of tools and methods, nor the present freedom of transfer of information as to what the other man is doing, and so did the best they could under the circumstances. Finally, the higher grade of the ores made the choice less vital.

The ever-increasing need to mine low-grade orebodies, with a fixed or even lower price for the metal produced, has caused an increasing need for low-cost mining. In most new mines this has simplified the problem of selection by eliminating from consideration all of the high-cost systems, but it has also complicated the choice by the necessity for refinement in detail and a close adjustment of the plan selected to all the conditions at the property. In these low-cost problems any textbook division of mining systems as applied to "beds, veins or masses" avails little as a guide. It should be realized at the outset that to be successful under modern competitive conditions a system of mining must be worked out with elaborate detail, with careful time scheduling and control of the various elemental operations, and that after it is introduced it must be run with a large amount of technical direction.

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* Assistant Professor of Mining, Yale University.

The factors that should be considered in selecting a mining system are numerous. Some few of them are so important that they alone will direct the choice of the scheme of attack on the orebody. Others affect only the detail of the work. Some may appear vital at the time the method is selected, but later events or further study may show that their importance has been exaggerated. Like most engineering design, the final selection must often be a compromise between conflicting influences. However, before an intelligent choice can be made, all the factors must be examined and their effect estimated. In the last analysis, the choice depends on over-all cost and safety.

The object of this paper is to present all of the variants that occur to the writer, with a brief word as to the place of each in the problem, in the hope that discussion or correspondence will fill any gaps and supply as many examples as possible of the influence of the less common conditions on actual choices of systems.

The factors in the accompanying table are divided for clarity into four main groups: (1) The internal factors, comprising those inherent in the orebody itself; (2) the external physical factors, or those of the terrain adjacent to the orebody; (3) the economic factors in the district; and (4) the economic factors of the industry or the particular product. Any discussion of these factors seriatim is difficult, since most of them are interrelated, and it is only when the variation from normal is extreme that many of them become important.

FACTORS INFLUENCING CHOICE AND DETAIL OF A MINING SYSTEM*

I. Internal Factors

- A. Orebody
  - 1. Shape and size
    - *a.* Horizontal area of commercial ore
    - *b.* Relation between length and width
    - *c.* Vertical extent
    - *d.* Regularity of boundaries and how they are determined
  - 2. Attitude
    - *a.* Dip and/or pitch
    - *b.* Relation to surface and adjoining public and private property
    - *c.* Relation to adjacent orebodies if any exist
  - 3. Continuity
    - *a.* Presence of oreshoots or of shoots of junore which may later be minable
    - *b.* Presence of large horses of waste
    - *c.* Presence of abnormally wide or narrow sections
- B. Ore
  - 1. Value per unit at mine
  - 2. Distribution of values
    - *a.* Uniform
    - *b.* Contains waste, high-grade ore, or ore of a different metallurgical nature that can economically be sorted underground

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* After publication of this paper as A.I.M.E. *Contribution* No. 21, Mr. Lucien Eaton suggested a number of valuable additions to this table, which have been made.

        - *c.* Different portions are of such a different character as to warrant selective mining or use of two or more systems
        - *d.* Ease of concentration
    - 3. Strength and large-scale structure influencing character of support required or behavior if caved
    - 4. Mineralogy and small-scale structure influencing
        - *a.* Breaking characteristics
            - (1) Explosive required
            - (2) Creation of rich fines
            - (3) Large pieces
            - (4) Behavior when crushed or undercut
        - *b.* Packing after long storage in stopes
        - *c.* Weathering, causing
            - (1) Swell
            - (2) Oxidation
        - *d.* Creation of health hazard due to siliceous gangue
        - *e.* Creation of explosion or fire hazard due to sulfide gangue
- C. Walls and capping
    - 1. Strength, affecting amount of support required, both local and as related to subsidence
    - 2. Action under pressure
    - 3. Breaking qualities, coarse or fine, free-running or sticky
    - 4. Regularity and dip (see Orebody above)

II. External Physical Factors

- A. The mining district
    - 1. Topography as it affects
        - *a.* Transportation
        - *b.* Shafts and plant location
        - *c.* Storage of ore and supplies
        - *d.* Disposal of waste
        - *e.* Drainage
    - 2. Underground water system
        - *a.* Source, quantity, quality and temperature
        - *b.* Water level
        - *c.* Presence of water-bearing strata
        - *d.* Presence of impervious strata
        - *e.* Seasonal variation
    - 3. Presence of faults and other main structural features
    - 4. Presence of lakes, rivers, or quicksand areas
    - 5. Temperature gradient and underground temperature
- B. Climate
    - 1. Seasons and variations of temperature
    - 2. Precipitation amount and seasonal variation
    - 3. Health conditions

III. Economic Factors in the District

- A. Labor
    - 1. Supply
    - 2. Ability
- B. Power
    - 1. Supply, regularity and cost of electric power
    - 2. Supply, quality and cost of fuel and boiler water

    3. Available water power
    4. Transmission lines
  C. Kinds and cost of materials available locally for underground support
  D. Character, use, and value of surface overlying ore
  E. Capital available as influencing
    1. Scale of operations
    2. Time when production must begin
  F. Geographical location as related to
    1. Transportation, kind and cost
    2. Distance to market for product
    3. Distance to source of supplies and equipment
  G. Location and availability of custom mills, repair shops and foundries

IV. Economic Factors in the Industry

  A. Regularity of demand for the products
  B. Range of unit price of the products
  C. Possible effect of operation being studied on price of the products and future demand

INTERNAL FACTORS

The Orebody

Shape.—The horizontal area of the ore is an important factor. Large areas increase the difficulty of supporting the ore in the back, and also mean that some method must be considered to support the whole hanging wall after mining, unless it may be allowed to cave. Large areas require increased attention to the lateral transport system. They offer, however, on each level many possible points of attack on the ore, and may permit the use of large-scale mining systems with large-scale ore-breaking (stoping) methods, frequently involving the use of bulldozing chambers and grizzlies. All of these statements can be reversed in referring to a small area. In addition, small areas indicate a short life for each level, and therefore that more levels must be opened each year for a given tonnage. The small tonnage available above each level means that the ore-loading arrangements must be simple and cheap, if the cost per ton passed is to be kept within reason. If the area is small and the length great, the question of minimum stope width and the possible use of resuing may be important. A small area prohibits a caving system. A narrow orebody also permits short pillars for wall support, and therefore a smaller percentage of ore locked up in them, since with ore of a given strength the length of the pillar determines the diameter required. In narrow orebodies stulls can be used to support slabs and to serve as working platforms for men instead of the timber structures or filling that might be required in a wider orebody. The horizontal area together with the relation between length and width determine the number and size of the sections into which the ore between two levels is to be divided for mining.

The vertical extent of the orebody affects the number of levels at which the orebody may be attacked. The vertical extent, horizontal area, and the relation of the orebody to the surface, are major points in deciding on the feasibility of open-cut work, or in determining the point of change from open-cut to underground mining. If the vertical extent is great enough, some underground work is indicated in spite of any other factors. Great vertical extent and steepness of dip also simplify the problem of lateral transport of ore, and the transport and placing of filling. A regular boundary favors plans in which easy transport down the footwall is essential, and is favorable for caving methods. If the boundaries are irregular, the recovery made by caving is lowered, and the system chosen should be a flexible one, such as square-setting, which can follow the ore anywhere. A marked difference in appearance and character between ore and wall rock reduces the amount of dilution and the profitless breaking of waste. Wall rock that is unavoidably mined may readily be sorted out. If the width of the ore is less than the minimum stope width and it parts freely from the walls, resuing is worth considering. Where the boundary is only a commercial one, care and close analytical control are required to maintain the grade of the ore hoisted. Moreover, if later industrial or technical changes may extend the boundaries of payable ore, a system permitting restoping of the walls would be advantageous.

Attitude.—Where the dip is flat, or where the pitch of an oreshoot is low even with a steep dip, gravity transport is more difficult and the ore must often be taken out of the stope by scrapers, conveyors, or cars. Except in thick orebodies, flat dips or pitches are unfavorable to the use of caving or other full stope systems. If the orebody is near the surface, open-cut work may be the solution. In a deep-seated orebody, caving may be possible even if the overlying surface is too valuable to destroy. Where other orebodies exist above the one under consideration, they must be mined first, if total cost is to be kept as low as possible. If they are not to be mined first, supported stopes must be used in the underlying orebody. The possible existence of parallel orebodies may so increase the exploratory value of openings driven in the walls to obtain filling that filled stopes would be preferable, even though other conditions point to another method of support.

Continuity.—If the ore occurs in shoots, the pitch of these shoots affects transport, and hence the mining system. If there is a probability that the vein material between the shoots may become commercial at a later date, it must be left in a minable condition. This low-grade portion will serve meanwhile to support the hanging wall, and thus allow, for example, delayed rather than contemporaneous filling. Where large horses of waste exist in the orebody, methods that will permit their being left unbroken or at least stored in the stope are desirable. If an orebody pinches and swells along either strike or dip, the system adopted must be flexible. Often the narrow portions may be left for support of the hanging wall. Sudden widening of the ore, unless anticipated and prepared for, may cause danger of caving.

The Ore

Value per Unit.—If the value per ton of the ore is great, a plan that permits a maximum percentage of recovery is essential. The loss due to leaving rich ore unbroken, left on the stope floor or buried in the filling, will overbalance any ordinary increase of cost due to more refined methods. For example, no caving system would be considered except top slicing, and support by unrecovered ore pillars would be out of the question. A low value per unit requires the adoption of a low-cost mining scheme, even if by it dilution is raised and recovery lowered. Where the value per unit is low and the overlying surface must be supported, pillar-and-chamber work with the pillars left in may be the only solution.

Distribution of Values.—If the metal is uniformly distributed through the orebody, no precautions are needed to insure regular head assays to the mill or smelter. If the orebody contains waste, high-grade, or ore of mixed metallurgical character, which can economically be sorted underground, a plan should be used that will allow this sorting to be done and will also allow the various products to be carried out of the stope separately, or to be stored in the stope. The possible loss of rich fines must be considered. Caving systems usually should be avoided. When the size of these diverse portions is great enough to make selective mining a possibility, a method where only small units are broken at a blast may be preferable; for example, square-setting. If the ore is easily sorted or concentrated there is less worry over a system that involves excessive dilution or unavoidable breaking of wall rock.

Strength and Large-scale Structure.—Strong ore allows larger areas of ore to be left unsupported during mining. When ore pillars are used for support, strong ore permits smaller pillars and therefore a greater percentage recovery. Weak ore usually necessitates prompt support, if men work under it; if the grade is high, this points toward square-setting; if the ore is low grade, a caving system may be the only one tolerated. If ore in different parts of the orebody varies in strength, safety requires methods suited to the weakest part.

Mineralogy and Small-scale Structure.—Where ore breaks readily, only small amounts of explosives are needed, and this means less shaking and shattering of the overlying ore or walls, a condition favorable to support. Small powder consumption also reduces the ventilation requirements. If the valuable mineral is brittle, protection against the loss of rich fines is needed. An ore that tends to break in large pieces may make local support of slabs necessary even though the stope back as a whole is strong. Under such conditions flat-backed stopes are preferable to rill or stepped-face stopes. Coarse-breaking ores usually require arrangements for blockholing, either in bulldozing chambers or in the stope itself. The ideal breaking characteristics for block caving comprise an ore that breaks into small pieces and a capping that breaks into large blocks. The reverse may prohibit block caving. If the ore tends to pack solidly, when stored for a time in the stope after breaking, the orebody is not suited to full-stope or block-caving systems. Ore that swells on weathering offers a serious problem of support. If the ore after breaking tends to oxidize or otherwise alter chemically, when subjected to the action of mine waters, mining plans demanding storage in the stope are proscribed. An ore with a siliceous gangue may need a method in which the suspension of dust in the air breathed by the miners is minimized. If the gangue is sulfide, there may be a dust explosion hazard, and a tendency to generate heat is created and must be coped with. When either of these dusts occur, they place a premium on layouts in which ventilation is easy. Continuous timbered stopes, whether filled or empty, and top slicing should be avoided with a sulfide gangue, because of the fire hazard.

Walls or Country Rock

Strength.—When the hanging-wall rock is strong, large areas of it can be left unsupported and open stopes are common. Strong walls are required for open underhand stopes. The strength of the wall over a long span will determine whether filling is needed either for immediate or ultimate support.

Action Under Pressure.—The way in which the hanging wall acts under pressure greatly influences the choice of the system. Walls that slab off always require some timber, if men are to work under them. A hanging wall that hangs up over a large area and then comes down suddenly is the most dangerous, and usually demands close filling. The easiest hanging to control is one that is flexible and comes down slowly and regularly. Such a roof permits longwall coal mining and similar systems in metal mines.

The Breaking Qualities of the wall rock when blasted affect the amount of dead work that may be permitted in rock openings. To mine an orebody having a weak hanging wall, a flat dip and a tough hard, irregular footwall is one of the most difficult problems to be solved.

Regularity and Dip.—The effects of regularity and dip of the walls are discussed above under similar terms as applied to the orebody.

EXTERNAL PHYSICAL FACTORS

Geology and Climate

An orebody forming part of a mountain mass will often be worked by open-cut methods, whereas the same deposit in a valley might be wholly mined underground. Large volumes of water encountered in a mine affect the mining system due to the action of the water on stored ore and also its effect on ore passes. A good example of the latter was seen by the writer at an iron mine in Michigan. Here the wet sticky ore dams up the ore passes, and holds back enough water to create a drowning risk for the loader at the chute gate below. Under such conditions the best plan was to remove the chute gate, and either use the ore pass only when a car was standing under it or else let the ore fall on to the floor of the level and clean it up periodically by scrapers. Much water in a mine, however, reduces the fire hazard and permits the safe use of timbered stopes even in sulfide ore. Where post-mineral faults exist, the problem of keeping with the ore is serious. Such faults make systems such as filled rill stopes, requiring a rigid layout, inadvisable, and lead to the selection of a more flexible plan. Faulted fragments may necessitate a variety of methods in the same mine. Faults almost always increase the difficulty of support and of systematic stope layout. Where lakes, rivers or quicksand areas occur near the orebody, special precautions must be taken to prevent their obtaining access to the workings. A securely supported stope is necessary, and caving the worst of practices unless the water can be removed. No chance should be taken even though the danger seems too far away to be considered. Good examples of the failure to realize the danger of such hazards were the Alaska-Treadwell disaster in 1917, and that at Barnes-Hecker in 1926. A great temperature gradient affects the choice of a mining system only in that it makes ease of ventilation more important. When orebodies are near recent volcanics, this ventilation consideration may be serious even in a small mine at a shallow depth.

Climate has a bearing in the choice between open-cut and underground methods. It also affects the cooling power of the intake air, and this influences the ventilation arrangements.

Economic Factors in the District

Labor.—A scanty or irregular labor supply increases the advantage of mechanized mining and of a mining scheme in which the tons per man-hour rate is high. Irregularity of the supply makes methods desirable in which the production rate can be cheaply changed. Skilled miners permit the safe and efficient operation of any system selected. Poor miners limit the range of selection to simple layouts, and in addition increase the cost. A plan should not be considered in which the available labor is incapable of being trained to function efficiently and safely.

Power.—If plentiful, regular and cheap, power will cut costs, especially in systems that can readily be mechanized. When power is not to be had, only simple hand methods may be considered. Electric power is particularly useful for ventilation.

Materials Locally Available for Support.—The kind and cost of these materials may directly determine the mining system, where support other than by ore pillars is necessary. Expensive timber and cheap filling materials may limit the choice regardless of other considerations, particularly if transportation to the district from outside sources is expensive. If filling is to be used, its possible sources in the district must all be considered and the laid-down cost estimated.

Overlying Surface.—If the overlying surface is not owned by the mine, or if it is too valuable to destroy, the ore must be removed by carefully supported underground mining. The possible damage to operations on or adjacent to the surface is definitely a part of the cost of mining. Sometimes lack of space for storing tailing, or the nuisance and expense of tailing piles near industry, will point to filled-stope methods. Unless the value of the surface above and near the orebody is lower than the saving to be effected, caving and open-cut workings should not be used.

Capital Available.—The capital available for the enterprise sets a limit beyond which the initial rate of exploitation cannot go, regardless of other considerations. When the initial capital is small, the operation must perhaps begin with the mining of high-grade portions by small-scale methods, at times with only hand labor. Large capital available at the start of mining permits the selection of the system best suited to the enterprise as determined by all the other factors; indeed, at times unlimited capital has induced the too rapid exploitation of the deposit. Only with plentiful capital for development and equipment can the large-scale, low-cost methods be selected. The scale of operations affects the number of working places that the mining scheme must provide, and the tonnage required per day from each. The demands of the capital raised also dictate the time at which exploitation should begin. Where the initial output is to be large and prompt, the mining plan demanded may be different from that where production may start on a small scale and work up to the maximum slowly. Some systems require long and elaborate preparation before actual mining can begin. The small number of mine superintendents who survive the period of gearing a new mine rapidly up to a large output is evidence of the difficulty of this problem.

Geographical Location of the Orebody.—When cheap and ample transportation is available, the cost of mining is lowered, and ore of a given grade is made more valuable. A greater variety of mining plans may then be considered than in isolated districts. The distance from the mine to a market for its product and from the source of supplies and equipment also vitally affects cost and profit, and therefore the allowable range of choice.

ECONOMIC FACTORS IN THE INDUSTRY

Regularity of Demand for Products.—When the rate of shipment to market is seasonal, owing either to a seasonal demand, as in the case of anthracite, or to a seasonal variation in transportation costs, as in the case of the Minnesota iron ores, a method that will permit shutdowns without excessive upkeep charges, and one that allows a marked and economical variation in rate of output, such as a full-stope system (shrinkage), has advantages.

Range of Unit Price of Products.—The range of the unit price affects the choice of a mining system in two ways. For the marginal producers a wide range in price is likely to make for intermittent operation and hence a desire for the scheme that will suffer least from an occasional stoppage of production. Even where the cost of the operation is well below the minimum price of the product, a marked change in price may cause an expansion or contraction of the workable limits of the orebody, and this must be considered in the selection.

Occasionally, when the potential output of the property is large in relation to the possible consumption of the product at a profitable price, this factor will limit the scale of the operation. An extreme case of this is diamond mining, where the allowed output is entirely a matter of agreement between producers and government.

Example of a Selection

As a clinical demonstration, let us attempt to select a mining system for an orebody. The Roan Antelope mine is in the public eye, and there are considerable published data on conditions there. The orebody is shaped like the bow of a blunt-nosed canoe, headed southeast. The north limb is steep, almost vertical; the south limb is flatter, having dips less than 50°. The bottom of the syncline—that is, the keel of the canoe—pitches about 10° to the northwest. The width of the portion to be mined first varies from 10 to 47 ft., and averages about 27 ft. The shape of the orebody is similar to that of the orebody of the New Jersey Zinc Co. at Franklin, N. J., though the thickness of the ore is less and the dimensions of the syncline are much greater. The length along the strike is 6 miles, but the north limb is missing for part of this distance. The ore is a banded, slaty shale, which stands fairly well, and drills and breaks readily. The ore averages over 3 per cent copper. The hanging wall is similar in character to and about as strong as the ore. The footwall is fairly regular, but not always strong. The ore is oxidized to a depth of 120 ft., and the oxidized ore possesses little if any value. Total reserves already developed are in excess of 100,000,000 tons. Good timber is scarce, and the labor supply is likely to be limited and to be subject to seasonal variation. The size and richness of the orebody and the capital available indicate the opportunity for a large output to start as soon as a treatment plant can be completed.

The easiest method of selection is to eliminate certain types of mining systems. Open-cut work is barred on account of the narrowness of the orebody and the depth to the top of the ore. Caving systems can also be eliminated, except possibly for the partial recovery of pillars, because the thickness of the ore is insufficient to justify the development required, and because they would cause too much dilution. Scarcity of suitable timber is a strong count against top slicing. Timbered stopes are not needed, if the men can be kept out from under a high back, and they would be prohibitively expensive, because of the poor timber supply. Contemporaneous filling is unnecessary, except where the river crosses the orebody, since the walls are fairly strong, and it would be costly where the dip is flat. As the surface is of little value, it is unnecessary to support it, and very little filling material is available. The tailings from the mill are not suitable for filling. When broken ore is left for a long time exposed to the air, there is considerable oxidation.

These conditions all lead us toward open stoping. The relatively thin orebody and fairly strong ore and walls point directly to pillar support, temporarily at least.

Having eliminated all the systems that appear inapplicable, let us now consider in detail the features desired:

1. The scheme adopted must produce a large output; since the orebody is narrow, this means many points of attack, or a large tonnage broken per shift at each point, or both.

2. The extraction must be fairly high, although the ore reserves are enormous, and, if pillars are to be left, the value of their contents must be overbalanced by a saving as compared to other supporting material, or else they must be recoverable at a later date.

3. The work must be simple enough for native miners under white supervisors, must be safe for ignorant labor, and must be capable of a high degree of standardization.

4. The wide variation in dip makes a system preferable that can readily be modified to suit any dip.

5. A probable seasonal shortage of labor makes desirable a high production from labor and the use of a method permitting a large amount of mechanization.

6. The varying labor supply places emphasis on the ease with which production rate may be adjusted without undue loss.

Summing up then, a supported-stope system of underground mining is indicated. Omitting for the moment the question of low percentage of recovery on first mining, a pillar-supported stope appears best, and will fulfill our other requirements. A full-stope (shrinkage) system, such as is used under somewhat similar conditions, except as to dip and width of ore, at the Homestake mine, might be considered, were it not for the oxidation of the broken ore in the stope and the ignorance of the common labor, but this would function only in the steep portions. The Beatson method is barred, because of the narrowness of the ore in the steep portions and because of the high cost of raising in Africa. Mineville practice of open pillar-supported stopes would be suited to the flat dips, if the hanging were strong enough, and the standard sublevel stoping system to the portions, where the slope of the footwall is sufficient to provide delivery of the ore by gravity. Gravity will take care of transport and loading on the steep dips, and gravity plus scrapers should solve the problem on the flatter slopes. If we support the back on ore pillars, we must either recover them or their value must be less than the cost of alternate methods of support. Large pillars are not needed, because temporary support only is necessary, and they can be partly robbed on the retreat, if we do not wait too long. It would be difficult to suggest any better method of mining this orebody than that proposed and pictured in the *Mining Magazine*, November, 1930.

DISCUSSION

(*Lucien Eaton presiding*)

R. PEELE,* New York, N. Y. (written discussion).—The subject of this paper is one of three that are closely related: Classification of ore deposits, classification of mining methods, and the selection of a mining method. Professor Warner discusses the last named of these three subjects from a somewhat new angle. Besides the questions concerning the nature and shape of the orebody, and the character of the ore and adjacent country rock, he introduces the local and general economic factors that may have a bearing on the selection of a suitable method.

It would seem that some of the factors listed in this paper have more to do with deposits occurring in the form of large masses and flat-lying thick beds than with steep-dipping veins. For the latter, the physical character of the ore and wall rock, the kind of metal, and the availability and cost of timber, are generally the determining factors, leaving comparatively little latitude in the choice of mining method. As a rule, consideration of these factors leads at once to the adoption of overhand or underhand work, by either open, shrinkage, or filled stopes.

On the other hand, for massive deposits having large lateral dimensions there is a wider range of choice, as between the numerous varieties of the caving and slicing systems, with their combinations. The choice of an appropriate method for such deposits is often much less obvious than for ordinary veins; because it is complicated by the greater need for considering the economic factors bearing on the problem, in addition to the physical characteristics of the orebody and its mode of occurrence.

C. F. JACKSON,† Washington, D. C.—About 1922, we thrashed this question out pretty thoroughly in the Institute and, as Secretary of the Mining Methods Committee, I had a small part in developing the system of classification that was finally adopted by the Institute and embodied in volume 72 of the TRANSACTIONS. We had a great deal of difficulty in arriving at a classification that was acceptable to everybody. As a matter of fact, we never did get such a classification. Perhaps one of the earliest classifications that came to my attention was one by Professor Sperr. That was a hobby of his for a great many years, and I think his first one must have come out at least 20 years ago. He had a very elaborate classification, which included 176 different mining systems.

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* Professor of Mining, School of Mines, Columbia University.

† Mining Division, U. S. Bureau of Mines.

It seems to me that the chief value of classification is for purposes of instruction of students. It is also helpful in clarifying our thinking along the line of mining methods.

The Bureau of Mines adopted a classification as a basis for a series of papers on mining methods and costs, and I had a part in setting up that classification. Having been through these discussions for several years, I can assure you that the simpler we can make the classification, the less difficulty we will get into and the less objection we will have to it. We based our classification on the method of support, which seems to me to be the prime factor in any mining method. It does not matter what we would like to do in handling a certain orebody, fundamentally the method that we use is controlled by the factors that determine the method of support that must be used while we are making the excavation. Charles Mitke used support as the basis for his classification. I am glad to see that Professor Warner has done the same thing, and I think that the latest classifications have all recognized the fact that, fundamentally, the mining system depends upon the method of support during the active life of the stope.

Professor Warner's paper sets forth the fundamental principles about as clearly as I have ever seen them put down on paper. I do not agree with one or two of his terms, like "roundabout stoping," which I question would be generally accepted by mining men, although it might come to be used. Otherwise I am heartily in accord with his classification.

Some have used the direction of stoping, sequence of stoping, method of handling, the type of deposit, and other factors as a basis for classification. There still seems to be some question as to the basis for correct classification, but I think in a general way we are coming to realize the fact that the method of support is the fundamental thing.

J. L. G. Weysser,* State College, Pa.—I should like to offer one point in support of Professor Warner's suggestion to keep coal-mining methods classified separately from metal-mining methods, if only because of the decided difference in terminology that exists. It has been my pleasure, I suppose I should say, to listen to a metal-mining man and a coal-mining man argue for two hours at a hard coal property. They were both talking about the same thing but using different words.

R. K. Warner (written discussion).—The truth of Professor Peele's statement is borne out by the fact that when the writer started to set up this paper he found it necessary to clarify his ideas by first working out the Classification of Mining Systems printed in *Mining and Metallurgy*, February, 1931, and the conception of the elements of mining systems printed in *Mining and Metallurgy*, March, 1932. These two articles should really form an integral part of the paper under discussion. The writer believes that the third step mentioned by Professor Peele—the classification of orebodies from the standpoint of mining methods—is not an impossible one, and that the material for such a classification exists in these papers. The paper attempts an exhaustive summary of the factors that must be considered in selecting a mining system and many of those cited become important only in those border-line cases where internal and external physical factors do not restrict the choice to a single system.

In reply to Mr. Jackson, the prime importance of the question of support was considered by the writer to be so obvious as not to require elaboration in the paper. It is given, of course, the first place among the elements of mining systems.

In addition to the above discussion the writer received a letter from Mr. Lucien Eaton, which comprises a detailed study of the paper. Most of Mr. Eaton's suggested changes have been made in the table on page 12. Mr. Eaton also makes two other criticisms, as follows:

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* Instructor in Mining, Pennsylvania State College.

"I do not think your paper lays enough emphasis on the comparison or balancing of the higher cost of filling methods against the possible dilution or mining loss and the loss of ore in pillars entailed by methods of mining that bring on subsidence or support the ground permanently.

"There is another point upon which I do not think that you have touched at all. This is the necessity at times of maintaining a definite grade of ore without regard to cost. This applies, of course, only to 'shipping ores' that are sold as mined. For instance iron ores in the Lake Superior district in order to be salable must be maintained at a certain minimum grade, regardless of the cost per unit of iron. Not only is there a heavy penalty imposed on ores that carry less than 48 per cent natural iron, but there is very little market for ores carrying less than 50 per cent iron natural, and none at all for leaner ores until the grade of siliceous ores is reached. These ores carry 35 to 40 per cent iron, and silica to correspond, and sell for about 75¢ a ton at the mine."

These points are both well taken and have a distinct bearing on the choice when a more expensive mining system which produces cleaner ore and better extraction must be balanced against a cheaper one lacking these advantages. The writer, in thinking of the cost of a mining method, includes in that cost the loss due to ore left unmined and that due to dilution. In the case cited dilution might easily prohibit such a system as sublevel caving and make top slicing the only feasible method.

Application of Principles of Similitude to Design of Mine Workings*

By P. B. Bucky† and A. L. Fentress,‡ New York, N. Y.

(New York Meeting, February, 1934)

The purpose of this paper is to present a scientific method for determining the proper span and shape of roof for safe and economical mining; at present, these two vital factors wherever pillars are left to support the roof are determined by local practice or general experience. To develop the method described here, experimental laboratory work was done. The arch adopted was a segment of a circle, a flat roof being considered an arch with an infinite radius. Hitherto, arches have been used underground without the background of stress analysis that generally precedes arch construction on the surface. In many cases results have not been as expected, because of the lack of knowledge of the factors affecting its strength and behavior.

A theoretical and experimental study of arches should furnish information that will make it possible to design safe and economical surface structures as well as underground openings. For example, on the subject of mine drifts and tunnels, the present idea is that arching their backs increases their strength; this paper will indicate that this is not always true, and if it can be shown that under certain conditions arching becomes unnecessary an economic saving will result. In room-and-pillar work a question may arise as to whether arching the roof and gobbing the excavated material will result in timber saving, and a decrease in the dangers from falls of roof. It is to be distinctly understood that each mine must be treated as an individual problem, and that answers to these questions and many others can now be found after experimental laboratory data have been obtained.

MINE-ARCH STRUCTURE

Fig. 1 shows the elements that affect the strength or behavior of the experimental arch. They are the span, crown thickness, radius of curvature, rise and restraint. The restraint in all cases was made sufficient to cause the arch to fail without displacing the restraining sides. Fig. 2*a* is a photograph of a model arch prepared for testing in the centrifuge. The model is of sandstone held in place in a steel holder by type metal.

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* Research project of Engineering Foundation and Columbia University. Manuscript received at the office of the Institute Nov. 21, 1933.

† Assistant Professor of Mining, Columbia University.

‡ Student, School of Mines, Columbia University.

The fundamentals of this investigation, conducted with models and applying the principles of similitude, have been described previously,[1] but may here be briefly summarized as follows. A scalar model of any

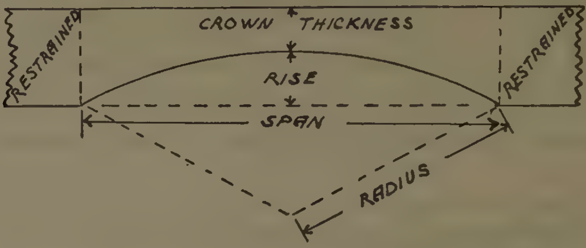

FIG. 1.—ARCH ELEMENTS OF UNLOADED ARCH UNDER INVESTIGATION.

weighty structure will behave like its prototype if the model and prototype material are the same, and if the effective model weight is increased in the same proportion as its linear model dimensions are diminished. The effect of an increase in weight is obtained by substituting a centrifugal field of force for the gravitational field, the model being placed in a suitably designed centrifuge.

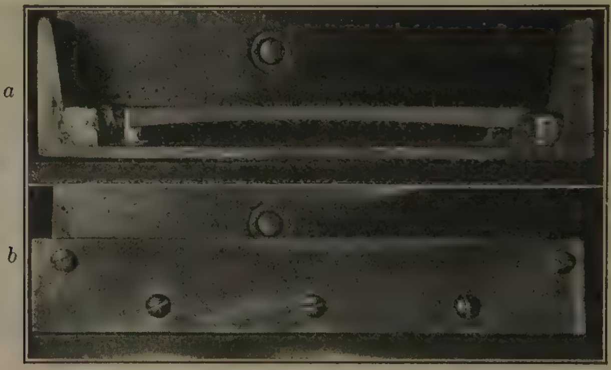

FIG. 2.—(*a*) MODEL ARCH IN ITS HOLDER; (*b*) JIG FOR PREPARATION OF MODEL.

It has just been stated that the factors affecting model strength and behavior are the span, rise or radius, crown thickness, and restraint. To determine the effect of each factor, all other factors must be kept constant. If the span and radius are constant, the rise must also be constant; for, from Fig. 1, with a given span or chord and radius, the rise or middle ordinate is fixed. Restraint can be made constant by extending the structure sufficiently beyond the arch curve and holding the model in place with type metal (Fig. 2*a*). The only remaining variable is crown thickness. By testing arches with different radii and varying the crown thickness of each, the effect of any single factor may be ascertained.

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[1] P. B. Bucky: The Use of Models for the Study of Mining Problems. A.I.M.E. *Tech. Pub.* 425 (1931).

The Use of Models under Centrifugal Force. *Trans.* Amer. Soc. Mech. Engrs.

The Effect of Approximately Vertical Cracks on the Behavior of Horizontally Lying Roof Strata. See page 212, this volume.

The models were cut from sandstone and limestone. Rectangular pieces approximately 8 in. long by 1 in. wide but varying in thickness were first cut. Each piece was placed in a steel jig (Fig. 2*b*) having the proper radius of arch. The arch was then cut to conform to the jig by emery wheels, files, and abrasives. The model was then set in the holder, both were placed in the centering and pouring device,[2] and the type metal was then poured. The model in its holder then appeared as in Fig. 2*a*.

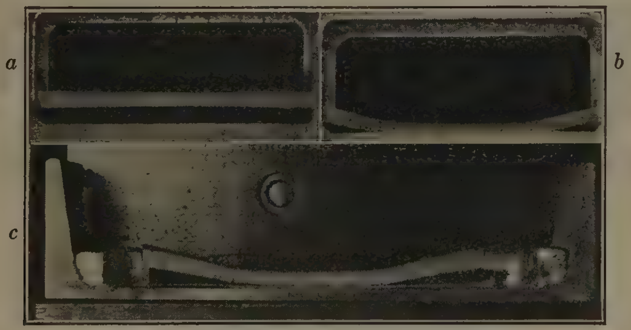

Fig. 3.—Models during and after test.
a. Rotating at 600 r.p.m.
b. Failing at 1229 r.p.m.
c. Failed model removed from centrifuge.

Test Procedure

After being set in the holder the model was balanced against a counterweight in the balancing device,[3] then model and counterweight were placed in the centrifuge and rotated. Observations on the behavior of the model while rotating were made photographically by a stroboscopic mercury arc light. Fig. 3*a* shows the appearance of a model while rotating at 600 r.p.m. and evidently standing up well. In Fig. 3*b* the same model was rotating at 1229 r.p.m. when failure occurred. Fig. 3*c* shows the model after failure and after being taken out of the centrifuge. At present it is possible to observe only a portion of a model while it is rotating. The rotating speeds were calculated from readings of a voltmeter in circuit with a generator connected to the centrifuge shaft through gearing; the generator is of the straight-line type and develops 6 volts per 1000 r.p.m. at the centrifuge.

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[2] Described in detail in third reference of footnote 1.

[3] Described in detail in third reference of footnote 1.

In the data sheet reproduced in Table 1, columns 1 to 6 contain information concerning the model, number and dimensions. Column 7 gives the voltmeter reading at failure of the model, and column 8 the speed of the centrifuge in r.p.m., calculated as follows:

R.p.m. = voltmeter reading × 1000 ÷ 6, since 6 volts = 1000 r.p.m. For example, a reading of 9.1 volts $= \frac{9.1 \times 1000}{6} = 1516$ r.p.m., or $1516/60 = 25.27$ r.p.s.

The calculated model ratio (column 9) is obtained from the formula M.R. $= \frac{4\pi^2 rn^2}{g}$, where M.R. = model ratio which is equal to either (1) the number of times the normal model weight that the model is loaded; or (2) the number by which the linear dimensions of the model must be multiplied to give similar linear dimensions of the prototype represented.

In this formula: $\pi = 3.1416$; r = radius of centrifuge in feet; i. e., the distance from center of rotation to center of gravity of model; n = r.p.s.; g = gravity, 32.2. For example, from Table 1, model 1 broke at 1516 r.p.m. The centrifuge radius was 8.56 in. (the distance from the center of rotation to the center of gravity of the model). The model ratio therefore $= \frac{4\pi^2 rn^2}{g}$,

or $$\frac{4 \times \pi^2 \times \frac{8.56}{12} \times \left(\frac{1516}{60}\right)^2}{32.2} = 556.$$

Experimental Results

The curves of Fig. 4 show the model ratio at which arch models with a given radius of curvature failed, plotted against the crown thickness. The data from which these curves were plotted are the same as those of Table 1, columns 3 and 9. Each curve represents a group of models of a definite radius, and, since the span is constant, the effect of rise is included and is also constant. The model ratio is plotted against crown thickness, which is the only variable. Following is the procedure for the 30-in. radius sandstone curve (Fig. 4), three models being run. Columns 3 and 9 of the Table 1 show that with a crown thickness of ⅛ in. model 4 failed at a model ratio of 181. This determines a point on the curve of which the ordinate is 181 and the abscissa ⅛ in. Model 5 had a crown thickness of ¼ in. and failure occurred at a model ratio of 637, which determined another point in the curve, of which the ordinate is 637 and the abscissa 0.25. Model 6 had a crown thickness of ⅜ in. and failed at a model ratio of 835, which determines a third point in the curve of which the ordinate is 835 and the abscissa 0.375. The present equipment does not allow the testing of models to failure when the crown thickness exceeds ⅜ in. The curves for the 40-in., 60-in., and infinity radii sandstone models were similarly plotted.

TABLE 1.—*Data Sheet for Sandstone Models*

Model Dimensions, In.						Experimental Data at Failure			Prototype Dimensions at Failure, Ft.					Safe Prototype Dimensions and Model Ratio					
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
Model No.	Radius	Crown Thickness	Rise	Over-all Depth	Span	Voltmeter Reading	R.p.m. Calculated	Calculated Model Ratio	Radius	Crown Thickness	Rise	Over-all Depth	Span	Radius	Crown Thickness	Rise	Over-all Depth	Span	Model Ratio
1	20	⅛	0.2730	0.3980	6½	9.10	1516	556	922	5.78	12.6	18.4	300	230	1.44	3.15	4.6	75	139
2	20	3⁄16	0.2730	0.4605	6½	11.1	1850	818	1357	12.67	18.6	31.3	442	339	3.16	4.65	7.8	110	204
3	20	¼	0.2730	0.5230	6½	12.0	2000	968	1606	20.13	21.8	42.0	523	401	5.03	5.45	10.5	130	242
4	30	⅛	0.1775	0.3025	6½	5.17	862	181	452	1.88	2.6	4.50	97.9	113	0.47	0.65	1.1	24	45
5	30	¼	0.1775	0.4275	6½	9.75	1625	637	1592	13.24	9.4	22.6	344.6	398	3.31	2.35	5.6	86	159
6	30	⅜	0.1775	0.5525	6½	11.2	1867	835	2087	26.00	12.3	38.4	451.7	521	6.50	3.07	9.6	112	209
7	40	⅛	0.1308	0.2558	6½	4.10	683	114	379	1.18	1.2	2.4	61.6	94	0.29	0.30	0.60	15	28
8	40	¼	0.1308	0.3808	6½	7.37	1229	363	1208	7.55	3.9	11.5	196.3	302	1.88	0.97	2.8	49	91
9	40	⅜	0.1308	0.5058	6½	10.0	1675	675	2247	21.00	7.3	28.4	365	561	5.25	1.82	7.1	91	169
10	60	¼	0.0449	0.2949	6½	4.25	708	122	610	2.53	0.45	2.9	66	102	0.63	0.11	0.72	16	30
11	60	⅜	0.0449	0.4199	6½	7.80	1300	409	2045	12.76	1.5	14.2	221	511	3.19	0.37	3.5	55	102
12	60	½	0.0449	0.5449	6½	10.35	1725	715	3575	29.31	3.1	32.4	386	893	7.32	0.77	8.1	96	179
13	∞	0.3025	0	0.3025	6½	4.20	700	118	∞	2.97	0	2.9	63.8	∞	0.74	0	0.72	16	29
14	∞	0.4275	0	0.4275	6½	7.37	1232	364	∞	12.95	0	12.9	196.9	∞	3.23	0	3.2	49	91
15	∞	0.5525	0	0.5525	6½	9.97	1662	663	∞	30.49	0	30.4	358.6	∞	7.62	0	7.6	90	166

Effect of Crown Thickness

In previous publications, it was shown that placing a model in a centrifugal field had the following effects:

1. The load on the model is increased in the same proportion as the model ratio is increased.

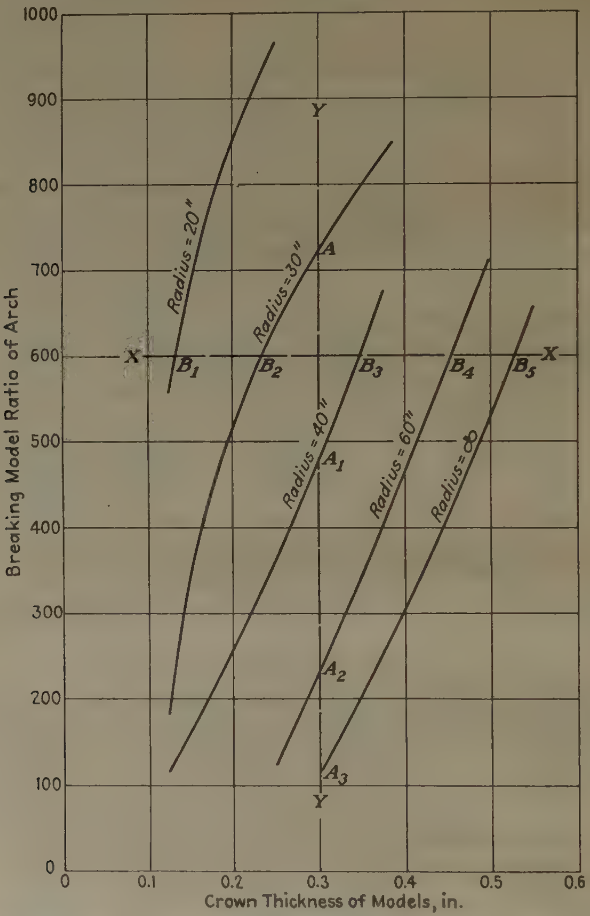

FIG. 4.—PERFORMANCE OF UNLOADED SANDSTONE MODELS. SPAN CONSTANT AT 6½ INCHES.

2. The model behaves similarly to a prototype of the same material, where the similar linear dimensions are equal to those of the model multiplied by the model ratio.

The fact that the model load is increased by centrifugal force provides a means of comparing the relative strength of differently shaped structures. For example, the 30-in. radius sandstone curve (Fig. 4) shows that a model with ⅛-in. crown failed when its load was 181 times its normal weight, while another model with ¼-in. crown failed when its load was 637 times its normal weight. It is evident, therefore, that the model with the ¼-in. crown is at least 3.52 times as strong as the one with the ⅛-in. crown thickness. This is true because the model with a ¼-in. crown weighs more. Increasing the crown thickness to ⅜ in. increased the load at which failure occurred to 835 times its normal weight. In these models the span, rise and radii were the same, the only variable being crown thickness. The 40-in., 60-in. and infinity radii curves of Fig. 4, plotted from data in Table 1, substantiate the results of the 30-in. radius curve here discussed. It may therefore be said that experimental evidence for the material, the type of structure, and within the limits tested, indicates that an increasing crown thickness increases the strength of this type of structure, the rate of increase being shown by the curves of Fig. 4.

Effect of Radius

If in Fig. 4 line YY is drawn parallel to the Y axis, this line cuts the 30-in., 40-in., 60-in. and infinite radii curves at points A, A_1, A_2 and A_3. It is evident also that the crown thicknesses of the models represented by the above points are all equal to 0.3 in. Point A_3 represents a model with an arch of infinite radius (i. e., a flat roof) which failed at a model ratio of 118; point A_2, a model with 60-in. radius arch which failed at a model ratio of 235; point A_1, a model with 40-in. radius arch which failed at a model ratio of 480; point A, a model with 30-in. radius arch, which failed at a model ratio of 726. Lines parallel to YY, drawn anywhere in Fig. 4 will show the same general results. These results show that *if crown thickness and span remain constant the load that this type of arch structure will carry increases as the radius decreases.*

For this comparison, all elements of the arch structure were kept constant except the radius or rise.

Arch Structures of Equivalent Strength and Span

In Fig. 4, if line XX is drawn parallel to the X axes, i. e. perpendicular to YY, the curves for the various models are intersected at points of which the breaking model ratios are the same; which means that the arch structures represented by points B_1, B_2, B_3, B_4 and B_5 will each fail when loaded to 600 times their weight. For example, the arch structures of Table 2 all have the same strength, the data being taken from Fig. 4. Since the crown thickness has an important bearing on the strength of this type of structure, the radius of arch to put in for a given crown thickness and span is determinable. From the column of over-all depths (Table 2), which is equal to crown thickness plus rise, it may be noted that the structures represented by point B_2 with an over-all height of 0.4125 in. and a 30-in. radius, has the same strength as B_3 with an over-all height of 0.4778 in. and a radius of 40 in.; B_4 with an over-all height of 0.4979 in.

TABLE 2.—*Arch Structures of Approximately Equal Strength*—600 × *Model Weight*

	Model Dimensions, In.				
Point	Span	Radius	Crown Thickness	Rise	Over-all Depth
B_2	6.5	30	0.235	0.1775	0.4125
B_3	6.5	40	0.347	0.1308	0.4778
B_4	6.5	60	0.453	0.0449	0.4979
B_5	6.5	∞	0.530	0.000	0.530

and a radius of 60 in.; B_5 with an over-all height of 0.53 in. and an infinite radius. The importance and economic application of this point, in determining level-pillar dimensions and shapes is shown later in the paper.

Prototype Dimensions for Failure

In discussing the curves of Fig. 4 on the basis that model behavior is similar to that of a prototype, it is found from the curve for 30-in. sandstone arch that a model with an arch of 30-in. radius, 6½-in. span, and ⅛-in. crown thickness fails at a model ratio of 181. That is, a prototype composed of the same material as the model will behave in the same manner as the model if its dimensions are as follows: span = (6½ × 181) ÷ 12 = 97.92 ft.; radius = (30 × 181) ÷ 12 = 452½ ft.; crown thickness = (⅛ × 181) ÷ 12 = 1.88 ft. From this curve, therefore, the dimensions of a prototype that will fail are obtained by multiplying the model dimensions by the model ratio at failure. These data for all models are entered in columns 10 to 14 of Table 1.

SAFETY FACTORS FOR MINE STRUCTURES

The data presented so far relate to models stressed to failure. An approach is now being made to the problem of determining from model structures that have failed the proper design of a safe structure. While failure of the structure is intended to occur in caving, slicing, longwall mining, and especially pillar extraction in coal, certain other methods demand safe structures; for example, the first stage of room-and-pillar coal mining, shrinkage stoping in ore, and various other systems that depend upon pillars and arched roofs for support.

In designing surface structures a safety factor of from 4 to 10 is used, the factor adopted being determined by the local building code.

If a model fails, it has evidently been stressed at its places of failure beyond its ultimate strength. Assuming a safety factor of 4, the problem is to determine the dimensions of the prototype that will allow maximum unit stresses one-quarter of those in the model at failure. In *Technical Publication* 425[4] it was shown: (1) that unit stresses at similar points in model and prototype are the same; (2) that the unit stresses at any point in a model vary as the model ratio. Hence, if a model fails at a model ratio of 1000 and its unit stresses at failure are equal to 4, a similar model rotated at a speed equivalent to a model ratio of 250 will have unit stresses of 1. That is, if the model breaks at a model ratio of 1000, the prototype will break if its dimensions are 1000 times those of similar linear model dimensions. The unit stresses in a prototype whose dimensions are 250 times those of similar model dimensions will be one-fourth those in the breaking model. Hence, with a safety factor of 4, the prototype dimensions should be equal to the model dimensions multiplied by one-fourth the model ratio at failure. If the same points of Fig. 4 are replotted to the safe model ratio (i. e., applying a safety factor of 4), the curves shown in Fig. 5 are obtained.

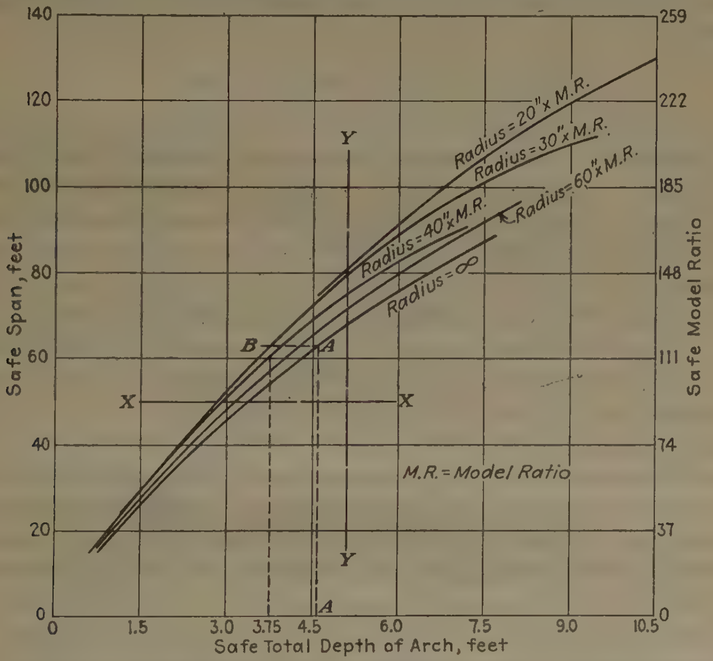

FIG. 5.—DIAGRAM FOR DESIGNING SAFE SANDSTONE ARCHES. SAFETY FACTOR OF 4.

The curves in Fig. 5 show the model ratios required to provide a safety factor of 4, and that will be as safe from a structural point of view as a surface building, the experimental data being those of Table 1. For example, the 30-in. sandstone curve with a crown thickness of ⅛ in fails at a model ratio of 181, which, for a prototype with a safety factor of

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[4] Reference of footnote 1.

4, becomes 181 ÷ 4, or 45.25 (Table 1, column 20). The radius of the safe prototype = 45.25 × (30 ÷ 12) = 113 ft. (column 15). Crown thickness = 45.25 × (⅛ ÷ 12) = 0.47 ft. (column 16). Rise = 45.25 × (0.1775 ÷ 12) = 0.65 ft. (column 17). Over-all depth is the sum of rise and crown thickness (column 18). Span of the safe prototype = 6.5 × (45.25 ÷ 12) = 24 ft. (column 19). In Fig. 5, safe model ratios and spans are plotted against total depth for an arch with a given radius of curvature. The model ratio equivalent of any span is determined thus: for a span of 20 ft. and a model with 6.5-in. span, the model ratio = (20 × 12) ÷ 6.5 = 37, which is plotted on the right-hand side of the curve.

Practical Applications

From the curves of Fig. 5 the following questions can now be answered. What radius of arch is required for the assumed material if the thickness is 3.75 ft. and the open span desired is 60 ft.? Follow the line marked "total depth" to point 3.75, above which the safe span for a flat roof (infinite radius) is found to be only 54 ft.; the roof therefore must be somewhat arched. On the 60 × M.R. curve the safe span is 57.5 ft. As this is insufficient, on looking further it is found that an arch with a 40 × M.R. radius will support safely a 60-ft. opening. At the right-hand side is the equivalent model ratio of 111; the radius of curvature of the arch is therefore 111 × (40 ÷ 12) or 370 ft.

It is evident that a set of these experimental curves for a given case will be useful in determining safe dimensions and shapes for mining structures, guesswork being replaced by scientific data. For example, when working a vertical deposit 50 ft. thick, by shrinkage stoping, the shape and dimensions of the level pillars may be determined in this manner. On Fig. 5 at the 50-ft. span mark draw line *XX*. This cuts the curves at points which determine the dimensions and shape of level pillars having approximately the same strength. Thus, the intersection of *XX* with the infinity curve shows that the level pillar may be 3.37 ft. thick if its under surface is flat. The intersection with the 60 × M.R. curve shows it may be arched to a 60 × (92.5 ÷ 12) = 462.5-ft. radius, the over-all thickness being then 3.15 ft. Similarly, the 40 × M.R. curve shows that if arched to a radius of 40 × (92.5 ÷ 12) = 308.3 ft., its over-all depth is only 3 ft. Thus, the ore tied up in level pillars may be reduced by adopting correct dimensions and shape without affecting strength.

Another general type of problem is the maximum economic span that may be obtained by arching. If line *YY* (Fig. 5) be drawn at a thickness of 5.1 ft., for example, a flat roof is seen to allow a safe span of 68 ft., while if the roof is arched to a radius of 30 × (148 ÷ 12) = 370 ft., the span that may be worked is 80 ft. The increased safe span with this arch is thus 12 ft., or about 18 per cent. The desirability of arching mine roofs being a question of economics, the effects are readily calculated. In ore mines worked by overhand stoping, especially in wide veins, a set of similar curves may be prepared for determining the shape of the back that should be used, and how this shape should change on approaching the level above.

APPLICATION TO SURFACE STRUCTURES

In Fig. 6 the safe curves for both sandstone and limestone are shown, based on the circular arch; similar curves for plain concrete, reinforced concrete, wood, or other material, could also be derived and, moreover, the shape of the arches could be changed to the parabola and catenary, giving additional groups of curves.

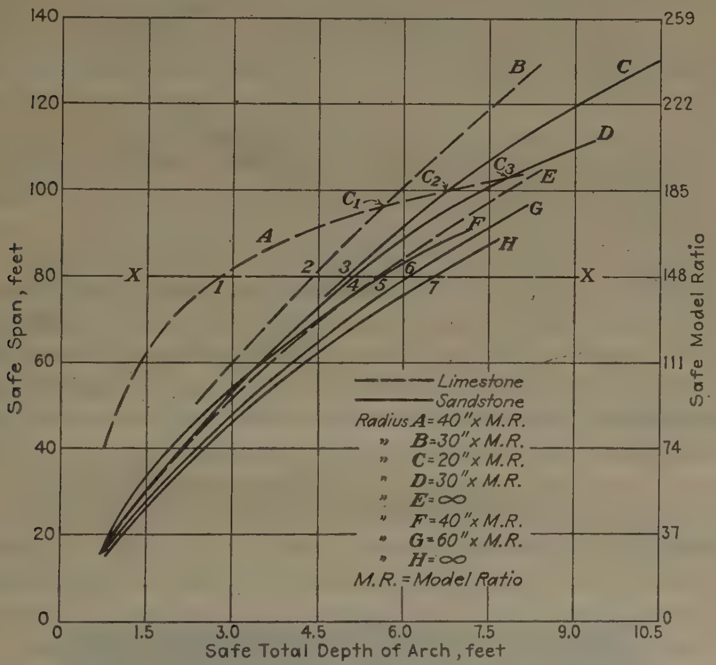

FIG. 6.—DIAGRAM FOR DESIGNING SAFE ARCHES IN SANDSTONE AND LIMESTONE. SAFETY FACTOR OF 4.

For example, assume that either sandstone or limestone may be chosen for an arch having an 80-ft. span and a safety factor of 4. By drawing line XX in Fig. 6 at the 80-ft. span, it is found that structures represented by the points 1 to 7 will satisfy safety requirements. Point 1 represents a limestone structure of which the radius is $(40 \div 12) \times 148 = 493$ ft., over-all height $= 2.85$ ft. and span $= 80$ ft. Point 7 represents a sandstone arch of infinite radius, 80 ft. span and 6.02 ft. thick. The dimensions and shapes of structures represented by the other points may be similarly determined, all having approximately equal strength. By computing the amount of material necessary for each structure, the minimum amounts and costs can be found.

In Fig. 6, the points c_1, c_2 and c_3 are those at which the curves for the limestone and sandstone structures cross one another, and represent structures of the same strength, span and over-all height, but with different arch radii. Thus the intersection at c_2 shows that a limestone structure with a 100-ft. span, an arch radius of (40 ÷ 12) × 185 = 616.6 ft., and an over-all depth of 6.75 ft., has the same strength as a sandstone structure with a 100-ft. span, radius of (20 ÷ 12) × 185 = 308.3 ft. and an over-all depth of 6.75 ft. Hence, the sandstone structure will take less material, and it may be noted that structures of weaker material, properly designed, may require less material for the same strength than if built of stronger material and improperly designed.

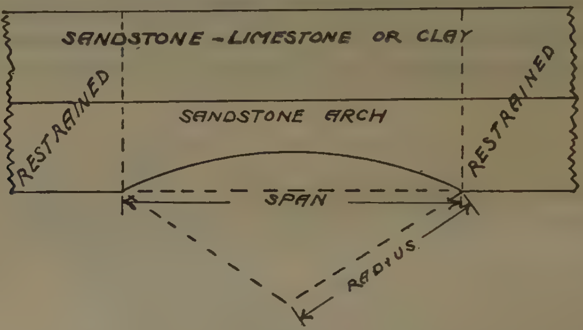

FIG. 7.—ARCH ELEMENTS OF LOADED ARCH UNDER INVESTIGATION.

LOADED ARCH STRUCTURES

The previous discussions relate to the type of arch shown in Fig. 1, which is important from a structural point of view and as the basis for studying more complicated designs, but is a type not often encountered in mining operations, except in steeply dipping orebodies. Most mining structures involve stratified layers overlying the material being mined, and in which the roof arch is made. In studying the behavior of such arches, loaded with different material, the experimental work was limited to a single stratum of varying composition and thickness (Figs. 7 and 8). In all of the tests, following the procedure already explained, the arch was

FIG. 8.—LOADED ARCH MODEL IN ITS HOLDER.

of sandstone while the overlying strata were sandstone, limestone or clay. Table 3 is the data sheet, prepared like Table 1 except that, since the model consisted of two layers and the centrifuge radius was small as compared with the model thickness, a correction factor was applied. For example, 0.25 in. of sandstone cover had an effective radius of 8.2 ÷ 8.5, or 96 per cent of that of the arch. The load due to the cover was therefore 0.96 × 0.25 in., or 0.24 in. (See columns 9, 10, 11 of Table 3.) The method of determining the data in the other columns has already been explained. Failure was assumed when the arch failed, not when the overlying material failed; arched mine roofs with thick covers fail before the cover does.

Model 1 (Table 3), the type of arch used in tests Nos. 14 and 1 to 7, inclusive, was not loaded and failed at a model ratio of 753. In models 2 to 7 the arch structure was the same as that of model 1, but the overlying materials and its thickness were varied. The experimental results should therefore give a means of determining the effects of overlying materials, of the kinds tested and of variable dimensions, on the behavior of the arch structure. The arch for models 8, 9 and 10 was of sandstone with the same over-all depth as model 1, but with a 40-in. radius. Models 11, 12 and 13 were flat beams with a thickness equal to an over-all depth of 0.5 inch.

The curves in Fig. 9, showing the strength of an arch structure as affected by the overlying material, were plotted from data in Table 3.

Point *C* shows that a sandstone arch of 30-in. radius, with no overlying material, failed at a model ratio of 753. The arch was then loaded with ¼ in. of sandstone, using no bond. Specimen 2 (Table 3) broke at a model ratio of 301; that is, the addition of ¼ in. of sandstone weakened the structure considerably. The corrected depth of cover (0.24 in.) is plotted against the failing model ratio. On loading the arch with ½ in. of sandstone (specimen 4), the break occurred at a model ratio of 441. As the thickness of sandstone overlying the arch was further increased, the arch apparently became stronger; for, with 1 in. of sandstone cover, as in specimen 4, the arch failed at a model ratio of 470. The thickness of the sandstone cover could not be increased beyond 1 in. with present equipment, but it is believed that a thickness of cover would be found that the arch would not fail until a model ratio of 753 is reached, i. e., until it is as strong as though there were no cover. This leads to the apparent paradox that the greater the thickness of cover on an arch the less the load on the structure.

As it was impossible to work with sandstone thicknesses of less than ¼ in., and since with no cover the arch broke at a model ratio of 753, the dashed line was drawn to show the approximate trend of this curve between points *C* and *I*. The reason for the greater loading of an arch by a thin cover becomes apparent on considering the deflection formulas.

TABLE 3.—*Data Sheet for Loaded Arch Models*

	Model Dimensions, In.										Experimental Data at Failure				Prototype Dimensions at Failure, Ft.						Safe Prototype Dimensions, Ft.							
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29
Model No.	Arch Material	Radius Arch	Crown Thickness	Rise Arch	Over-all Depth Arch	Cover Material	Over-all Depth Cover	Radius C.g. Arch	Radius C.g. Cover	Corrected Depth Cover	Voltmeter Reading	Calculated R.p.m.	Model Ratio Arch	Model Ratio Cover	Radius Arch	Crown Thickness	Rise Arch	Over-all Depth Arch	Over-all Depth Cover	Span	Radius Arch	Crown Thickness Arch	Rise Arch	Over-all Depth Arch	Over-all Depth Cover	Span	Model Ratio Arch	Model Ratio Cover
1	SS.	30	0.325	0.177	0.50	No cover	No cover	8.50	No cover	No cover			753	No cover	1882	20.33	11.29	31.62	0	407	470	5.08	2.82	7.90	0	101	188	No cover
2	SS.	30	0.325	0.177	0.50	SS[a]	0.25	8.50	8.12	0.24	6.70	1117	301	296	752	8.12	4.21	12.33	6.15	162	188	2.03	1.05	3.08	1.53	40	75	74
3	SS.	30	0.325	0.177	0.50	SS	0.50	8.50	8.00	0.47	8.10	1350	441	415	1102	11.90	6.17	18.07	17.01	238	275	2.97	1.54	4.51	4.25	59	110	103
4	SS.	30	0.325	0.177	0.50	SS	1.00	8.50	7.75	0.91	8.35	1391	470	428	1175	12.69	6.58	19.27	35.52	253	293	3.17	1.64	4.81	8.88	63	117	107
5	SS.	30	0.325	0.177	0.50	LS[b]	0.25	8.50	8.12	0.24	7.40	1233	367	350	917	10.30	5.13	15.04	7.00	198	229	2.57	1.28	3.76	1.75	49	91	87
6	SS.	30	0.325	0.177	0.50	LS	0.50	8.50	8.00	0.47	8.65	1443	501	471	1252	13.52	7.01	20.54	19.31	270	313	3.38	1.75	5.13	4.82	67	125	117
7	SS.	30	0.325	0.177	0.50	LS	1.00	8.50	7.75	0.91	8.90	1483	532	485	1330	14.36	7.44	21.81	40.25	287	332	3.59	1.86	5.45	10.06	72	133	121
8	SS.	40	0.369	0.131	0.50	LS	0.25	8.50	8.12	0.24	7.35	1225	363	347	1198	11.14	3.95	14.88	6.94	196	299	2.78	0.98	3.72	1.73	49	90	86
9	SS.	40	0.369	0.131	0.50	LS	0.50	8.50	8.00	0.47	8.40	1400	472	444	1557	14.49	5.14	19.35	18.20	255	389	3.62	1.28	4.88	4.55	63	118	111
10	SS.	40	0.369	0.131	0.50	LS	0.75	8.50	7.87	0.70	8.65	1443	501	464	1653	15.38	5.46	20.54	28.76	270	413	3.84	1.36	5.13	7.19	67	125	116
11	SS.	∞	0.50	0	0.50	LS	0.25	8.50	8.12	0.24	7.00	1167	328	314	∞	13.44	0	13.44	6.28	177	∞	3.33	0	3.36	1.57	44	82	78
12	SS.	∞	0.50	0	0.50	LS	0.50	8.50	8.00	0.47	8.20	1367	450	424	∞	18.45	0	18.45	17.38	243	∞	4.61	0	4.61	4.34	60	112	106
13	SS.	∞	0.50	0	0.50	LS	1.00	8.50	7.75	0.91	8.25	1376	456	416	∞	18.69	0	18.69	34.52	246	∞	4.67	0	4.68	8.63	61	114	104
14	SS.	30	0.325	0.177	0.50	Clay	0.50	8.50	8.00	0.47	6.80	1133	308	282	790	8.31	4.62	12.93	11.84	176	192	2.07	1.15	3.23	2.96	44	77	70

[a] Sandstone.
[b] Limestone.

As overlying material is in effect a beam, it deflects according to the beam formulas, and the arch loading is probably proportional to the beam deflection. The formula for beam deflection is $K\frac{WL^4}{ED^2}$, in which the constant K depends on the type of beam, W = weight per unit volume, L = length, E = modulus of elasticity, and D = depth (in this case, depth of cover). Since L is also constant for all structures here considered, the deflection varies inversely as the square of the depth and the modulus of elasticity; that is, as the thickness of cover increases, the deflection and load on the arch are decreased.

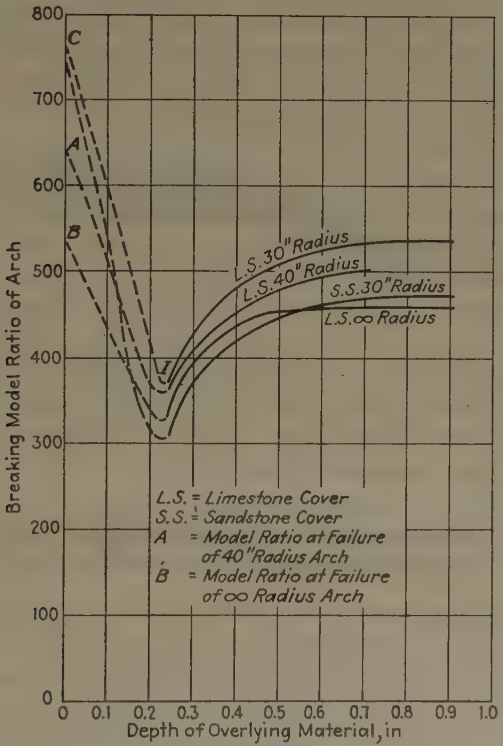

FIG. 9.—PERFORMANCE OF LOADED ARCH MODELS, ALL OF SANDSTONE, WITH SANDSTONE OR LIMESTONE COVERS.

Taking now a cover material, like clay, whose modulus of elasticity is very low (specimen 14, Table 3), with a cover 0.5 in. thick, the arch structure failed at a model ratio of 282; while with a 0.5-in. sandstone cover (specimen 3) it failed at a model ratio of 441. As the clay loaded the structure much more than the sandstone, it follows that the less the modulus of elasticity of the material overlying the arch the greater the load on the structure, all other factors being the same.

For the limestone curves of Fig. 9, models 5 to 13, inclusive, were tested (see Table 3). The arch structure for Nos. 5, 6 and 7 was the same as in the previous models, and the curve was constructed like that for the 30-in. sandstone. The modulus of elasticity of limestone being greater than that of sandstone or clay, it will evidently load the arch structure less, as is shown by the curves. With ¼-in. sandstone cover, the arch failed at a model ratio of 301; with ¼-in. limestone at 367. With 1 in. of sandstone, failure occurred at a model ratio of 470; with 1 in. of limestone at 532.

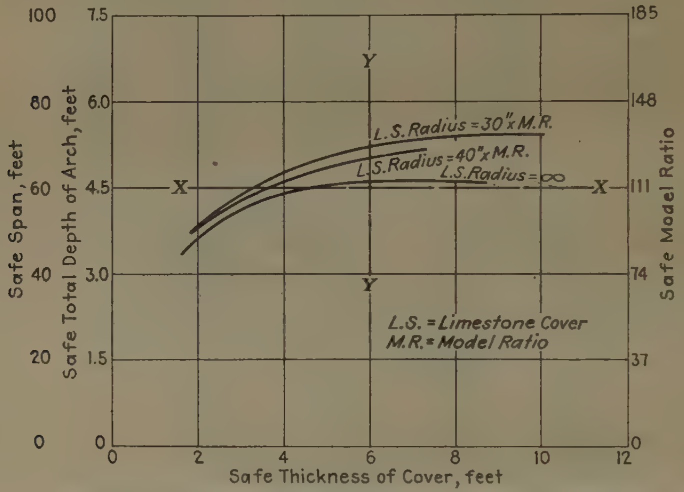

FIG. 10.—DIAGRAM FOR DESIGNING SAFE LOADED ARCHES IN SANDSTONE, WITH LIMESTONE COVER IN EACH CASE.

The foregoing results show that the load on an arch structure depends on the thickness and modulus of elasticity of the material immediately overlying the arch. With very thin covers, and between certain limits, the load on the arch increases with the thickness of the cover. (See dotted portions of curves.) Beyond a definite limit, the load on the arch structure decreases as the thickness of cover increases. Overlying materials whose modulus of elasticity is low will increase the load on an arch structure.

PRACTICAL APPLICATIONS

Considering the safe design curves in Fig. 10, and assuming workings with a cover of 6 ft. of limestone, underlain by a variable thickness of sandstone forming the roof, certain questions arise: (1) What spans can be safely worked? (2) What should be the shape of the roof?

By drawing through the curves the line *YY*, corresponding to 6 ft. of safe cover, it is seen that: (*a*) The intersection of *YY* with the limestone infinite-radius curve shows that, if the roof is sandstone, flat and 4.57 ft. thick, the safe workable span is 61 ft. (*b*) The intersection of *YY* with the 40 × M.R. curve shows that, with a sandstone roof 5.02 ft. thick and arched with a curve whose radius is 124 × (40 ÷ 12) = 413.3 ft., the workable span is 67 ft. (*c*) The intersection of *YY* with the 30 × M.R. curve shows that, with sandstone roof 5.25 ft. thick and arched with a curve whose radius is (30 ÷ 12) × 130 = 325 ft., the safe workable span is 70 feet.

If a 60-ft. span is assumed as necessary in this material, what radius of arch will be required for different thickness of roof and cover? Draw line *XX* at the 60-ft. mark (Fig. 10) and note its intersections with the various curves. The intersection with the 30 × M.R. curve shows that with a limestone cover 3.2 ft. thick, underlain by 4.5 ft. of sandstone, and arched with a curve of (30 ÷ 12) × 111 = 277 ft. radius, a 60-ft. span will be safe. The intersection with the 40 × M.R. curve shows that with a limestone cover 3.6 ft. thick, underlain by 4.5 ft. of sandstone, and arched with a curve of (40 ÷ 12) × 111 = 370 ft. radius, a 60-ft. span will be safe. The intersection with the curve of infinite radius shows that with a limestone cover of 4.5 ft., underlain by 4.5 ft. of sandstone, and with a flat roof, a 60-ft. span will also be safe.

Other interesting information is obtainable by a study of Fig. 10 in connection with Fig. 5. For example, from Fig. 10 a flat sandstone roof 4.57 ft. thick will support a 61-ft. span, if the overlying limestone is 6 ft. thick or more. If in Fig. 5 the line *AA* be drawn vertically from the 4.57-ft. mark to intersect the infinite radius curve, and then a horizontal line *AB* be drawn to intersect the other curves, it is found that other roof structures will allow the same spans since they are all of equal strength. Thus, the intersection of line *AB* with the 30 × M.R. curve shows that the sandstone roof may be 3.75 ft. thick, and, if arched at a radius of 116 × (30 ÷ 12) = 290 ft., it will support the same span as a flat roof 4.57 ft. thick. The intersection of *AB* with the 40 × M.R. curve similarly shows that the sandstone roof may be 3.97 ft. thick and if arched at a radius of (40 ÷ 12) × 116 = 386.6 ft. it will support the same span as either of the others. Hence, by means of these curves the shape of the roof in different portions of a mine, where roof and cover thickness are varied, can be determined to secure safety with a given span. If the roof shape is kept constant while the thickness of roof and its cover vary, the safe workable span can also be obtained.

SUMMARY AND CONCLUSIONS

The conclusions drawn here apply only to the materials and types of structures specifically mentioned in this paper.

1. Certain types of mine structures may be designed from experimental data and be as safe as surface structures.

2. The general method presented for the design of mine structures may be applied to determine the safe and economic design of surface structures.

3. The solution of practical mining problems, respecting safe underground spans, roof shapes and the shape and size of level pillars is suggested.

4. The experimental work checks the following known facts: (*a*) Increasing the crown thickness of an arch structure increases its strength and the open span that may be worked under it; (*b*) decreasing the radius of curvature of an arch structure increases its strength and the open span that may be worked.

5. The rates of increase accompanying conditions 4*a* and 4*b* are shown to be obtainable by experiment.

6. The load due to strata overlying an arch structure decreases as the thickness and modulus of elasticity of the overlying material increases, except in the case of small thicknesses of cover. The rates of increase and decrease are obtainable experimentally and are shown by curves.

The experimental results obtained with the present apparatus necessarily apply to small structures. New apparatus is being built to deal with problems involving depths up to 3000 feet.

In conclusion, the author of this paper desires to emphasize: first, that the practical application of the suggested method necessitates a rather complete geological knowledge of the strata involved; second, that the theory and the experimental data presented should not be applied by inexperienced engineers; third, that both theory and data must be considered in the light of the senior author's previous publications on the subject, *Technical Publication* No. 425 and *Contribution* No. 4 (see p. 212, this volume) published by the American Institute of Mining and Metallurgical Engineers.

DISCUSSION

(*Lucien Eaton presiding*)

L. Eaton,[*] New York, N. Y.—There have been a good many comments on the use of models and there is a difference of opinion as to their value. I think the difficulty lies in two points primarily, one being the lack of homogeneity of materials underground and the other the fact that underground we are working in solid geometry and not in plane geometry. That is another field that Professor Bucky can investigate whenever he feels like it. It has infinite possibilities.

P. B. Bucky (written discussion).—Individuals may find it difficult to accept model solutions of mining problems, because of the lack of homogeneity of underground materials and the impression that solid geometry (three-dimensional) problems are not solvable by these methods. The writer feels that the real difficulty, however, lies in the inertia to the acceptance of new concepts concerning underground structures by the profession. It is difficult for individuals accustomed to solving their problems in a certain manner to accept new methods. In the past the mining engineer in dealing with underground problems has made little use of his scientific background in mathematics, physics and mechanics, with the result that when asked to recall these fundamentals after a period of 10 to 20 years he finds it difficult and trying to do so.

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[*] Consulting Mining Engineer.

The effects of nonhomogeneity of materials in the prototype having been discussed from the standpoint of small and large aggregates and cementing materials.[5] It is here shown that these effects are ascertainable and that they define limits of accuracy. The determination of the effect of cracks has been undertaken and partial results are available.[6]

It is desired to correct an erroneous impression that this method of testing applies only to plane geometry (two-dimensional) problems and not to solid geometry (three-dimensional) problems. This impression may be due to the problems selected for discussion. Many of the mining problems are two-dimensional ones; i. e. problems whose solution is dependent on the relationship between two dimensions in the model; for example, the determination of tunnel or adit shape and dimensions, or the determination of room span in a coal mine whose length is to be several times its span. A model to solve either of these problems would consist of the underlying material, ribs or walls, an opening and overlying material. The relationships of importance would be between the opening dimension (i. e., width and height), and thickness of material; the effect of the dimensions at right angles to these is negligible. In Fig. 11 the dimensions of importance in a two-dimensional problem are *A* and *B*; *C* may be anything.

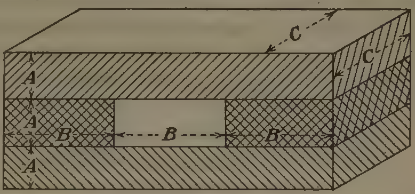

FIG. 11.

A three-dimensional problem is one where three model dimensions are important. This condition exists when the horizontal dimensions of an opening are nearly the same. For example, what are the maximum dimensions of a square or round opening to be made underground? For the solution of this type of problem the dimension *C* becomes as important as the dimensions *A* and *B*. This type of problem, which is much more difficult of solution by the application of present-day mechanics, lends itself admirably to a solution by models.

B. F. TILLSON,* Upper Montclair, N. J.—May I express the thanks of this meeting to Professor Bucky for having done two things. First, he has brought to us his experimental work before he has the last word to say about it, when too many of us wait until we think there is nothing more to be said before we speak. The second thing is that he has given us some knowledge as to the application of the experimental results he has obtained, and procedure for applying them.

G. S. RICE,† Washington, D. C.—I would like to ask a question about the arrangement for testing the strength of the beam in the model. Apparently the beam is rigidly restrained on the ends by type metal?

P. B. BUCKY.—Yes.

G. S. RICE.—That being so, are not the values for strength much higher than would be obtained for a natural beam or slab of sandstone or slate in a mine roof?

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[5] First reference of footnote 1.

[6] See page 212.

* Consulting Engineer.

† Chief Mining Engineer, U. S. Bureau of Mines.

The strength figures impress me as being unduly high. Tests made on concrete stoppings in a concrete tunnel at the U. S. Bureau of Standards, Washington, and on a larger scale in the U. S. Bureau of Mines experimental mine, showed that the breaking strength of a uniformly loaded unreinforced concrete beam, restrained on the ends, varied with the degree of compressibility of the restraining buttresses. Natural bedded rock strata usually have at intervals joint crevices or slip planes more or less normal to the bedding. Under lateral high pressure these crevices close, and this movement plus the compressibility of the rock permits breaking under uniform load at a lower pressure than if there was rigid restraint.

Surely a natural beam or slab of stone like a roof stratum would behave under vertical load in a manner similar to an unreinforced concrete beam or slab of the same thickness and with ends having similar degrees of restraint. In the tests mentioned above, as the load on a beam was increased to the rupturing point the first effect was a tension break across the bottom layer at the middle, as such a material, like stone, has low tensile strength. The break then extended upward as the restraining walls or buttresses compressed until only a thin layer at the top was under compression. The beam as a whole was then acting like a flat arch. When the unit compressive strength of the top layer was exceeded, total failure resulted.

It would be interesting if Professor Bucky should make a test in which cross-fractured pieces of the same material used in the beam were inserted at the ends of the beam to duplicate the condition that I have cited of a mining condition.

P. B. Bucky (written discussion).—We do not know whether the values for strength that we obtained are much higher than would be obtained for a natural beam or slab of sandstone or slate in a mine roof. A set of conditions may be assumed and a line of reasoning followed that may result in an answer of either yes or no. If conditions as to the mine are fully specified—that is, the geology and samples of the material furnished—a model may be determined and a value for strength obtained that will be correct within experimental limits.

Mr. Rice further raises the important question of the relationship between beam restraint and load; i. e., the relative load per unit length that a beam can carry without being stressed to the breaking point. The two general types of beam to be considered are: (1) the simple beam—that is, the beam with free movement at its two supports—and (2) the perfectly restrained beam—one so firmly held at the supports that even with the beam loaded to failure there is no movement at the supports. It is known that the unit load a beam can carry increases to a definite limit as the restraint is increased. If the restraint is increased beyond this point, the unit load the beam can support does not increase but remains constant. The effect[7] of slip planes whose sides are not in juxtaposition is to lessen the unit load that will produce failure in the beam. Since the beams tested and discussed here had no slip planes, it is reasonable to expect that they shall be stronger. While in general it is assumed that slip planes affect the strength of mine roofs, it is the senior author's experience that where they are present and the sides are in juxtaposition the strength of the roof is affected very little.[8] Mr. Rice's statement of the manner in which the beams he referred to failed are evidence of the presence of very little restraint. Fully restrained beams fail as follows: (1) The first cracks that occur are on the top of the beam over the supports. This is to be expected because the unit tensile stresses at these points are twice those at the bottom center. (2) The underground tension crack in the mine roof appears later.

It is difficult to decide as to the relative strengths of concrete and sandstone roofs. The experiments carried on assumed a definite sandstone-roof condition without the presence of cracks or slip planes, and the models were made accordingly.

[7,8] See page 226.

R. D. HALL,* New York, N. Y. (written discussion).—The authors of this paper declare (p 25): "The present idea is that arching the back (of mine drifts and tunnels) increases their strength; this paper will indicate that this is not always true, and if it can be shown that under certain conditions arching becomes unnecessary, an economic saving will result. In room-and-pillar work, a question may arise as to whether arching the roof and gobbing the excavated material will result in timber saving and a decrease in the dangers from falls of roof."

I have never heard that anyone advocated arching the roof to increase its strength as a whole; such arching would be too expensive. It merely protects the worker against breakage of the soffit, or underside of the roof, which is subject to deterioration from acid waters in the roof, or which is so loosely connected to the roof structure and so laminated or rotten that it has strength inadequate in itself to carry its own load. As a rule, the underside of the roof parts from the stiffer roof above, and hence carries none of the load of those superior measures.

Let me make a comparison with domestic conditions. The home owner rarely fears that the ceiling lacks adequate strength, but is afraid that plaster which has become wetted may fracture and fall, for it has, perhaps, expanded already and is bulging down. The plaster represents what the English term the "immediate roof"; the ceiling is the solid structure above it. The former is analogous to the drawslate, or back, and the latter is analogous to the main roof.

That immediate roof of the mine may contain soluble materials that will weaken it. It may contain bodies like calcium carbonate that may turn to sulfates and expand it. There are many such bodies. In that event, the immediate roof will expand vertically and try to lift the entire cover. It will also expand laterally into the drift, room or tunnel. When it expands laterally, it may find the expansion is opposed and that the only relief, as with the plaster on the ceiling, is to be obtained by bending, or bulging, the laminas of rock downward into the opening, and this movement nearly always results in speedy breakage. Mine managers who have been disciplining men with discharge for not breaking down such blisters before they break down on them will be slow in accepting the assurance of the authors that experiments have shown that drawslate, or back, will not fall and kill employees. They will say that it happens at the mines, if not in the laboratory.

Some engineers say that if the opening is wide, the expansion usually gives less trouble. By making an arch, the laminations in the measures being cut off can expand without buckling or blistering, merely sliding like cards over one another's surfaces without end resistance. The expansions are in three directions, but it seems the measures finding a direction of low resistance expand in that direction many times as much as if they expanded in three directions, and so satisfy their needs. Otherwise, the haunches of the arches would be ripped by vertical expansions, and the roof would be ripped along the rib line on either side. By this conversion of the expansion in three directions into one in a single direction toward an arched opening, the strain is released. At the same time, the acid waters are drained out, and the expansion is decreased.

At the top of the arch, the width is zero and the exposure also, and the top has plenty of support from the encroaching sides. So the arch relieves the lower layers from a strain they would be unable to carry. The purpose of the arch, which cuts only into the immediate roof, is not to supply resistance to vertical pressures transmitted from the superincumbent roof.

The conditions of the immediate roof, its weakening by chemical action and its lack of adhesion to the roof proper, are absent in the experiments made by the authors, and these experiments, therefore, are applicable only to conditions where the roof is not wracked by expansions and is a unit. Mr. Bucky's conditions are obviously special. If they were not, they would show preliminary weakness in the soffit or in the underside of the opening. Instead the arch breaks down completely and not on the underside only. No amount of experiment, no mathematical theory of similitude, no assurance that conditions are perfectly analogous can convince us who know that the underside of the roof often collapses without complete collapse of the roof that our observation is wrong and that interior collapse of the arch does not, at times, occur before collapse of the roof itself, which final collapse may never occur and may occur only after a very remote time.

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* Engineering Editor, *Coal Age.*

The authors in their summary declare: "The experimental work checks the following known facts: (*a*) Increasing the crown thickness increases its strength and the open span that can be worked under it," etc. What is proved is that, if the depth of arch from crown to supports is increased as much as the crown thickness is increased, the strength of the arch is augmented, but the figures show conclusively that if the depth of arch from crown to supports, in other words, the "cover" of the mineral deposit, remains the same, the roof is strengthened by the creation of an arch within it. That is, the roof is made more resistant to complete collapse by arching, given a certain depth of cover, and it has been seen how by arching the roof is protected against partial disintegration dangerous to life. Increase in depth of cover always assists in increasing the ultimate strength of the roof, and from this article one learns that caving within the span still further increases the strength.

Let us review the authors' figures as taken from Table 1.

Model	Crown Thickness, In.	Rise of Arch, In.	Over-all Depth, In.	R.p.m. at Failure
4 (arched slab)	0.1250	0.1775	0.3025	862
13 (unarched slab)	0.3025	0	0.3025	700
5 (arched slab)	0.2500	0.1775	0.4275	1825
14 (unarched slab)	0.4275	0	0.4275	1232
6 (arched slab)	0.3750	0.1775	0.5525	1867
15 (unarched slab)	0.5525	0	0.5525	1662

In every one of these cases, it takes more revolutions to break the arched slab than the unarched slab, the depth of the slab being constant. So arching the roof does strengthen it against complete collapse, though that is not at all the reason why the backs of mine drifts and tunnels should be arched. It is an important consideration where large areas have been extracted in mines, for it prevents roof from breaking which for safety should break.

The fact that falls of the soffit increase roof strength is the source of much trouble, as the roof that does not collapse, owing to the excessive roof strength, causes "bumps." On the other hand, an arched roof will not throw weight as far as an unarched roof. In all cases, in this paper, I use the expression "arch" in its popular connotation as cover with a curved soffit and do not mean merely the creation of an arch of stresses in cover that may still be unbroken.

Exception must be taken to holding the "restraint" by type metal. The material tested should be longer by far than the authors use and rest freely on its supports if not at its ends. The length of the "restraint" is a matter for careful study and experiment, as it is vital to correctness of results, and if wrongly afforded may vitiate all the conclusions.

I must add that with a span of 6¼ in. the arch is being revolved about a radius of only 8.56 in. Thus the ends are on a radius of 9.11 in. and under a proportionately greater stress. If possible the radius should be increased, so as to bring all parts of the arch under more equal strain.

P. B. Bucky (written discussion).—It is true that there has been little advocacy for the arching of entries or rooms in bituminous mines. In the ore mines and anthracite fields, the arching of gangways, adits and tunnels for increased strength has long been general practice. The strength of the arch as a structural device has long been recognized for surface structures and for railroad and water-supply tunnel work. In the senior author's apprenticeship in the Wisconsin zinc fields arching was practiced in the stopes.

The reason arches for underground stopes or room structures have not been discussed more in the past is that there were no definite means of determining how much arching was required or the cost and savings entailed thereby. In this paper means are given for determining: (1) whether arching will decrease the amount of artificial support; (2) the general shape and arch dimensions necessary; with the consequent ability to determine the amount of material to be taken out and the cost thereof.

It therefore follows from 1 and 2 that when arching will result in (*a*) a saving in the cost of support greater than the cost of making the arch, or (*b*) an increase in income from an orebody greater than the cost of arching, it is desirable.

Mr. Hall evidently is speaking from personal experience with coal-mining conditions where slate and thin layers of weak materials overlie coal measures. The senior author has had a similar acquaintance and in general agrees with Mr. Hall's observations as to that type of roof behavior, though it is difficult to accept his reasoning and evidence presented therefor.

Laboratory experiments presented as movies at the 1933 A.I.M.E. meeting, and at present available, show that a mine roof may separate from a stiffer structure above it because of its dimensions, or physical properties. Definite proof is available for the effect of these factors. Some of the other factors presented by Mr. Hall may have the effects suggested but more conclusive evidence is necessary.

The writing of this paper was undertaken from the standpoint of showing how tests may be run, and data obtained, plotted, and interpreted, for materials peculiar to any type of mining or locality. The authors, realizing that numerous factors may or may not have an effect on the behavior of a mining structure, devised means for determining the effects of each of these factors singly and in combination. This implies that the paper had for its aim the careful control of all factors so that discussion could be limited to them. To further specify conditions, the authors (p. 41) make the following statement: "The conclusions drawn here apply only to the materials and type of structures specifically mentioned in this paper." Since it is evident that Mr. Hall had different structural roof material in mind, his reasons are not clear for making the statements regarding "Mine managers who have been disciplining men" etc. on page 45. The authors cannot see by what further reasoning Mr. Hall concludes from the data in this paper that "caving within the span still further increases its strength."

For the conditions tested, the restraint required was such that any fracturing that occurred took place between the model support and the restraint without movement of the restraining element. This was accomplished. Attention is again called to the fact that the rotating model is placed with its long dimension parallel to the axis of rotation. The ends and center therefore revolve at the same radius. The only corrections to be applied are those for model thickness.

C. T. HOLLAND,* Lookout, W. Va. (written discussion).—The authors are to be congratulated upon their progressive methods of research on the structures composing mine roofs. The use of a centrifugal field to secure dynamic similarity between a model and its prototype gets around almost insurmountable physical difficulties to which other theoretically possible methods are subject. Even the method using the centrifugal field is subject to certain inaccuracies which as Mr. Bucky shows in a previous publication[9] can be made so small as to be practically negligible by making the model small relative to the radius of the centrifuge. This method seems to have a great disadvantage, however, because it will require models that are small, and to a certain extent it will rule out some of the problems that need investigation and that probably would yield valuable results to an investigation making use of a model. The problems of subsidence over coal mines and stresses in coal-mine pillars are of this type.

The results obtained in this paper are interesting and it would seem are valuable to those who are interested in the arch for structural purposes. Possibly the authors could have made the paper more valuable by telling more about the physical properties of the sandstone and limestones on which they experimented, especially the elastic properties of the rocks and strength in compression. When, however, an attempt is made to use models to investigate and design the distance to be spanned by arches cut into the natural rocks or beams and slabs composed of the rocks making up the mine roof, real obstacles are often encountered. It seems that these difficulties may be classed under the following heads: (1) mechanical difficulties in constructing a model to represent the prototype; (2) uncertainty as to the materials making up the mine roof; (3) uncertainty as to the stresses existing in the rocks from geologic changes which have occurred in the past and which possibly are occurring at the present; (4) inability to anticipate and make allowance in the model for stresses set up and physical changes made in the rocks by temperature and moisture changes acting over considerable periods of time.

Under the first heading, the difficulty would present itself of constructing to a small linear scale a model that would represent the joint planes and stratification plans common in all sedimentary rock. These will certainly have a profound influence upon the action of a mine roof. In addition to this, frequently mine roofs (particularly over coal seams) are made up of several thin strata of materials, each stratum having entirely different physical characteristics from the others. Moreover each layer may thin, disappear, reappear and thicken in a comparatively short distance in a most perplexing manner. A typical illustration is given in Fig. 12.[10] One can easily see that constructing a model to represent a condition of this kind on a small scale would be difficult to say the least. If, on the other hand, models were constructed on a scale large enough so that these conditions could be represented adequately, it would be difficult to construct a centrifuge large enough in which to test them, so that errors introduced peculiar to this method of research would not render the results practically valueless.

Under the second heading, uncertainty as to the material making up the roof, there are several factors to be considered. Among these would be: (1) variation in distance between joint planes from place to place, (2) variations in thickness of each stratum of different materials making up the roof, and (3) variations in physical properties of the materials composing the roof from place to place. Some information could be obtained by boreholes, but if the roof were highly variable, as shown in Fig. 1, we could never be quite sure just what our model should represent. Certain assumptions based on experience might be made but that of course is just what is done now, and there seems to be little reason to believe that a model solution would give any better result than our present rule-of-thumb methods.

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* Assistant Manager, Lookout Smokeless Coal Co.

[9] First reference of footnote 1.

[10] L. M. Morris: Roof of the Pittsburgh Coal Bed in Northern West Virginia. *Trans.* A.I.M.E. (1931) **94**, 151–158.

The third heading, uncertainty as to the stresses that may have been set up by geologic changes, would present a very difficult problem with which to cope. This factor would almost certainly have to be considered if the results are to be accurate. That in some cases such stresses would exist does not seem to be open to reasonable doubt, but just how they are to be detected and their intensities measured is a problem which at present does not appear to be solvable. Yet if a model is to be truly similar to its prototype these stresses would have to be reproduced in the model before it is subjected to test.

Under the fourth heading, inability to anticipate and make allowance in the model for stresses set up and physical changes in the rock due to temperature and moisture changes acting over considerable periods of time, factors present themselves that may not be of importance in some rocks. But in some cases these are real factors in the cause of failures of mine roofs. Recently it was recognized to be of sufficient importance at one particular mine to cause the management of the mine to install and operate an air-conditioning plant during certain seasons of the year.[11] It is difficult to see how a span for a roof affected in this manner could be successfully determined by the use of models.

It might be said in conclusion that the purpose of this discussion is not to condemn the use of models in mine design, but to point out some of the obstacles in the way of their use, which it seems will have to be overcome before their use will become general or before any great reliance can be placed upon results obtained from them for this purpose.

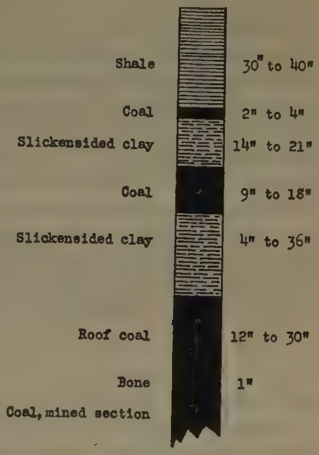

FIG. 12.

P. B. BUCKY (written discussion).—Mr. Holland's first objection is that the smallness of the model makes the use of this method inapplicable to problems of subsidence and stresses in coal-mine pillars. It is true that[12] the smaller the model in relation to the centrifuge diameter, the more accurate the results. Present models and equipment give results accurate within 10 per cent, which means that if model tests show that a break will occur when the pillars are drawn back 90 ft. the break probably will occur at 99. The extra 9 ft., which is the possible error of the machine, must be watched carefully. Mr. Holland evidently appreciates that it is more difficult to tell the position of a break within 10 per cent than it is to design a structure with a safety factor of 10. At the present time it is believed that methods of testing that will produce results foretelling within 10 per cent matters pertaining to subsidence and breaks are an advance. The percentage error may be reduced by applying corrections to results or by the use of larger centrifuges and models.

There is no reason why models cannot be made larger. The new centrifuge has a 4-ft. instead of 16-in. diameter and will take correspondingly larger models. Since problems of subsidence are in most cases reducible to the two-dimensional status, a 4-ft. centrifuge with models 1 ft. deep, 6 in. wide and 15 in. long rotated at speeds equivalent to a model ratio of 3000 will give subsidence and underground effects of large areas. The behavior of underground support has been ascertained with the centrifuge, and movies of the mine roof and support behavior for varying conditions are available.

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[11] J. H. Fletcher and S. M. Cassidy: Air Cooling to Prevent Falls of Roof Rock. *Trans.* A.I.M.E. (1931) **94**, 31–32.

[12] First reference of footnote 1.

For the solution and study of problems dealing with stress distribution in pillars, the laboratory has recently combined the photoelastic with the centrifugal methods of testing, whereby one passes polarized light through a bakelite model and produces a fringe pattern, which gives the value of the stresses and their distribution in the pillar. It is hoped that these results may be published in the near future.

The objections raised by Mr. Holland under the heading of mechanical difficulties in model construction are natural to one who has given the subject thought. The difficulties presented are not insurmountable if one applies the fundamentals of mechanics and structures in the construction of the model for test. Let us take his illustration, and see how one would construct a model[13] for determining the safe roof span, where roof coal is left and no artificial support is used. The worst conditions of roof to be met for the above problem, assuming it is sound engineering to design for that, may be listed as follows: 12 in. of roof coal overlain by 36 in. of slickensided clay, overlain by 9 in. of coal, overlain by 21 in. of clay, overlain by 2 in. of coal overlain by 30 in. of shale. The first procedure is to get samples of the roof and determine the characteristics of the various materials. Assume material characteristics such that the clay loads the coal and that the shale is fairly stiff and does not load underlying material. The procedure in model making would be as follows: (1) Assume everything below shale and to roof coal as clay (total, 68 in.). Make a model as follows. If the coal roof in model is assumed as $\frac{1}{2}$ in., clay $= \frac{68}{12} \times \frac{1}{2} = 2.8+$ in. Then test, obtain data, and draw curves as shown in this paper. Material is at present at hand, and being prepared for publication, that will discuss problems of this nature in more detail.

It is desired to emphasize: (1) that results are not perfect; (2) that the answers obtained by intelligently constructed and tested model are more nearly correct than mere guesses.

The question raised under the heading of uncertainty of material making up the roof has, in the main, been answered in the foregoing. For the effects of joint planes and the distances between them model tests will serve to give an index. Tests run and published have shown in general that joint planes with sides in juxtaposition or which have been filled have very little effect on rock strength.

If material varies in different parts of the mine it will be necessary to sample at those parts and probably use rooms and pillars of different dimensions or possibly different methods of mining in those sections.

Under the heading of stresses set up by geologic changes, conditions are conceivable where these factors might be difficult to cope with. The bituminous fields fortunately offer very few. Since the model material is taken from the mine, and since it is put under the same unit pressures while being tested, it is reasonable to expect that results will not be affected to any great extent.

Temperature effects will be included if the temperature of the model while rotating is kept the same as that of the prototype.

At present centrifugal model tests are not recommended to obtain moisture effects over a given time. The effect of these elements can be obtained more cheaply and effectively by present chemical means.

In appreciation of Mr. Holland's closing paragraph, it is desired to again point out that this method of testing is not to be considered as a cure-all and is not applicable to the solution of all mining problems, but is to be considered as a step in a direction that will encourage those in the profession to think about the problems presenting themselves, the obstacles to be overcome in solving them, and the possible methods of overcoming these obstacles.

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[13] See page 212.

Mining Methods and Costs at the Washington Mine of the Alan Wood Mining Company

By C. H. Loux,* Oxford, N. J.

(New York Meeting, February, 1933)

This paper deals primarily with the advantages realized from the adoption of methods more suitable than those previously used. The improvements noted include: (1) modification of practice in shrinkage stoping, (2) use of sublevel caving over grizzlies, (3) improved haulage and loading, (4) use of contract wages wherever possible.

The Washington mine is on Jonestown Hill, one mile south of Oxford, in northwestern New Jersey. It is producing 16,000 tons of concentrates a month, which are being shipped to the Alan Wood Steel Company's furnaces at Swedeland, Pennsylvania.

History

The first settlement in this region is reported to have been in 1735, and the magnetite ores were recognized immediately. The first furnace, built in 1741, began operation in 1743. During the Revolutionary War the output was chiefly used for making cannon balls, which, owing to the sympathies of the community, without doubt found their way into the hands of the British. This old furnace is still standing.

Several important veins were soon developed, including the Martin, Strayley, McKinley, Adelaide, Welsh and Lanning. Mining was carried to a depth corresponding to the capacity of the pumps available. When these became inadequate, the workings were abandoned, and another vein was opened up. The Washington vein was probably discovered about 1850, and the first shaft started in 1854. In 1879, William H. Scranton made the first complete magnetic survey of the district and serious exploration of the deposit was begun. Rather indifferent success attended the earlier efforts of the many companies that attempted to mine and smelt the ore. In 1929 the property was leased by the Warren Foundry and Pipe Co. to the present operators, and the modification of practice then inaugurated is described in this paper.

Geology

These veins, like most eastern deposits of magnetite, are formed from, or are contemporaneous with, pegmatite intrusions into a gneissic country rock. The ore therefore follows the banded structure of the gneiss, producing more or less lenticular shaped veins. The gneiss is usually fine grained, with bands of pink spar, accompanied by small amounts of hornblende and biotite. The ore is coarse, granular, massive magnetite, of metamorphic origin, containing primary apatite, quartz, pyrrhotite and actinolite in varying amounts, with some secondary pyrite and calcite. The principal impurities are apatite and quartz, aggregating approximately 10 per cent with about 2 per cent of sulfur in the form of pyrite.

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* Superintendent, Oxford Division, Alan Wood Mining Co.

In the Washington vein the orebody strikes N. 25° W. The dip to the east is 69° in the upper levels and gradually flattens to 45° in the lower levels. The orebody pitches to the south at an angle of about 50°. Several rolls in the walls occur, which were probably caused by pressure of pegmatite from the footwall side while the ore was still plastic. At these points the pegmatite partly, and in places almost wholly, replaces the ore, but does not penetrate into the hanging wall. The vein walls are distinct and well defined. There is no magnetite in the adjacent country rock and no apparent variation in iron content across the width of the vein. In the upper levels the vein was about 1200 ft. long and averaged 12 ft. in width. It has gradually shortened and widened with depth. The levels recently opened up, fourteenth and fifteenth, have a length of 600 ft. and an average width of 30 ft., with a maximum width of 80 feet.

PHYSICAL CHARACTERISTICS

The ore is hard and abrasive; it is readily brought down by blasting, but overbreak is in the form of large flat blocks which require much blockholing, breaking with hand hammers being impossible. These blocks, when loosened in the stopes, though very difficult to bar down, are of course just as dangerous to work under. To overcome the natural tendency of the men to be careless in barring down is a serious problem from a safety standpoint.

The country rock is also hard, abrasive, and is so tough that it will not shatter with moderate charges of powder. Considerable experience is required to "break bottom" on a cut in this rock, and to adjust the amount of burden placed on each hole so as to obtain good results.

DEVELOPMENT AND MINING PRIOR TO 1929

The first openings were flat slopes at about 25° to 30°. The first shaft of importance was sunk vertically through 100 ft. of sand, clay and broken rock to the capping of the vein, where the angle was changed to conform to the footwall. This was followed, with many changes in dip, to the tenth level. Pillars were left to protect the shaft but as mining progressed the pillars proved insufficient for support. After several bad caves the shaft was abandoned.

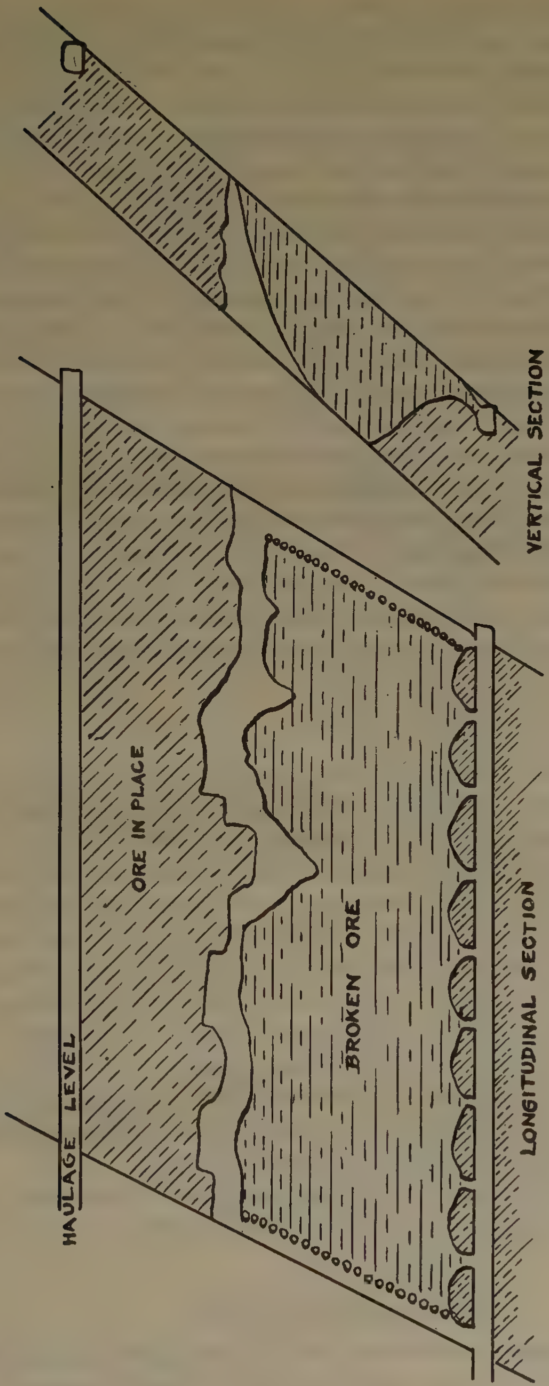


FIG. 1.—SHRINKAGE STOPING PRIOR TO 1929, WASHINGTON MINE.

A new shaft, sunk in 1917, is 8 by 19 ft. in the clear, and timbered for one manway and two hoisting compartments. It has an underlay of 69° throughout its entire length. It was 40 ft. in the footwall at the upper levels, but with the flattening of the vein diverged further from the footwall, and on the last level, the fifteenth, this distance had increased to 300 ft. Crosscuts were driven from shaft to vein. A drift was then run, following the footwall to the north and south limits of the ore. Chute raises, at 30-ft. intervals, were driven on 35° slope for 10 ft.; then branched, and connected one with another on as flat a slope as possible. These branches were finally coned out and undercut to the hanging wall.

A shrinkage system of mining was used (Fig. 1) drilling with hand-rotated stopers vertical holes in the back of the stope. The whole level was mined as one stope, access to which was through a cribbed manway at each end, which was timbered as mining progressed. As no raises were driven to the levels above until the stope was completed, ventilation was poor. The ore was slabby and dangerous to work under. It was difficult to do sufficient blockholing in the stope, and to draw the broken ore from the low, small, flat chutes. At this time the dip, on the lowest level, the thirteenth, was 47°, and the wide vein made shrinkage mining very difficult. Tramming was costly, owing to blocking of the chutes by large pieces and to the narrow, crooked drift following each bend of the footwall.

Alan Wood Mining Company

The company leased the Washington mine in 1929, and work of remodelling began in October of that year. The shaft was sunk 240 ft., and pockets were cut for the fourteenth and fifteenth levels. Meanwhile a storage sump was cut on the twelfth level to reduce pumping from three shifts to one shift. The drift on the thirteenth level was straightened and widened, and new chutes were installed wherever the character of the ground would permit. Stope preparation having been completed on the thirteenth level in 1923, shrinkage stoping was continued on this level. Wherever possible the old chutes were elevated and enlarged and underslung arc gates installed. New 24-in. gage track of 40-lb. rail was laid in the straightened drift. Easton 70-cu. ft. automatic, side-dump cars replaced the old gable-bottom, manually operated cars previously used. The arc gates are operated by air cylinders.

The surface plant was overhauled, the mill completely remodelled, and a standard-gage railway built connecting the D. L. & W. R. R. with the mine.

Mining in 1931

Haulage levels are driven on line in rock, approximately 20 ft. in the footwall and 200 ft. apart along the dip. Manway raises connect these

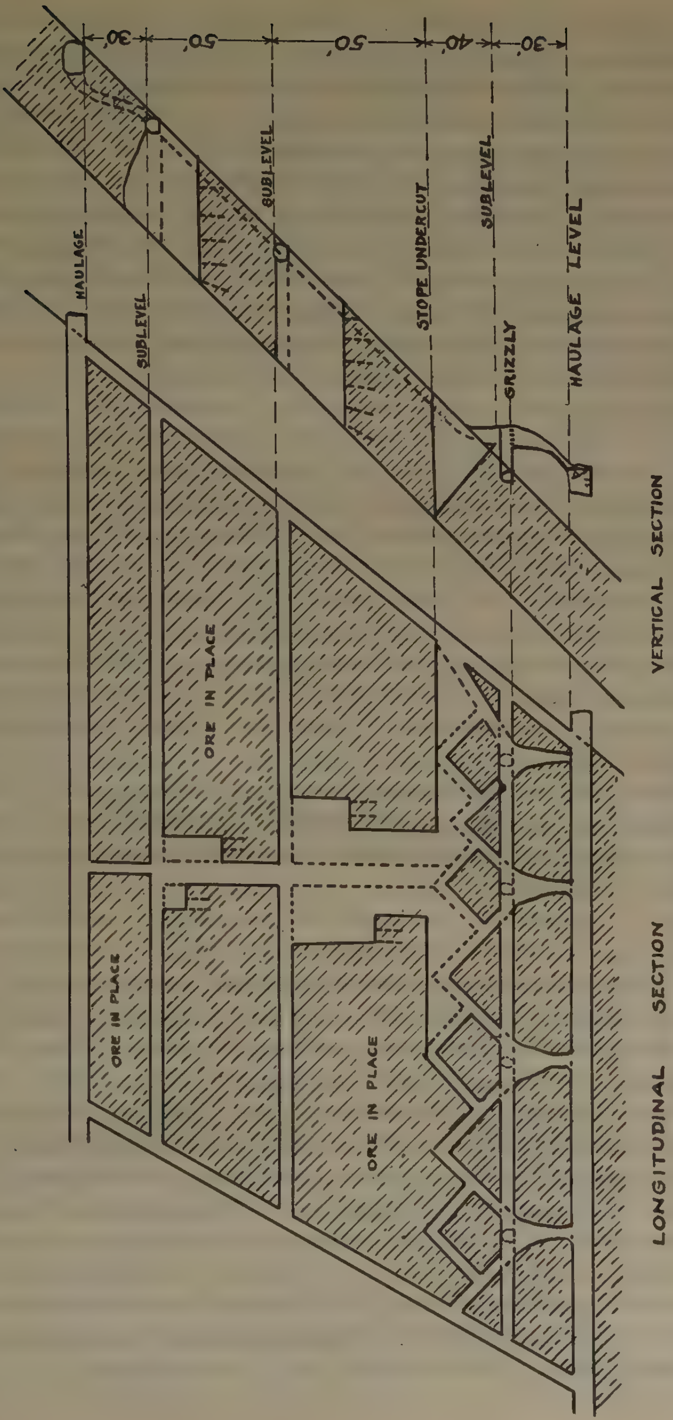


FIG. 2.—SUBLEVEL CAVING BY ALAN WOOD MINING CO., WASHINGTON MINE, 1931.

levels at each end of the orebody. Three sublevels, the entire length of the level, connect with the manway raises. The first sublevel is 30 ft. above the haulage drift, the next 120 ft. above and the last 170 ft. above; all are in ore and driven along the footwall. Chute-raises are on 80-ft. centers in rock and are coned out for storage and for grizzly chambers. These chambers connect with the lowest sublevel at its central point with a short crosscut. From both ends of the grizzly chambers, raises are run at 45° until they connect. These raises break through the footwall into ore on the second round, and then follow the footwall in ore. Connecting raises are also driven over each grizzly chamber in ore, and the whole stope is coned out and undercut to the hanging wall, leaving grizzly chambers with drawing points at each end. (Fig. 2.)

A raise is then driven from the back of the central connecting raise to the level above, and undercut from foot to hanging as far as the upper sublevel. Benches are cut at the intersection of the sublevels with this central raise, from foot to hanging. The benches are about 10 ft. wide, and mounted drifters are used to drill the rounds. In stoping, jackhammers are used to drill vertical down holes in two rows across the bench to a depth of 10 ft. These holes are fired with electric delay fuses. As the benches are carried downward, men with equipment and materials have access along the flat footwall from the nearest sublevel above. The bench is carried down until it breaks through, when the process of sidewiping a new bench is repeated. The labor is contracted: three drillers and a nipper make one crew and are paid on a "tonnage-broken" basis.

In the old stope on the thirteenth level, where shrinkage stoping had started, mining practice was modified with good results (Fig. 1). Because of the extremely wide vein, the broken ore when drawn from the chutes pulled on the hanging wall and lay on the footwall. To overcome this the footwall was undercut the length of the stope, with drifters. Flat holes were then drilled from this breast toward the hanging wall and blasted. The broken ore was used to set up on, to drill another breast, until the hanging wall was reached. The broken ore was then drawn out until the footwall was clean enough to repeat the undercutting. To drill the flat holes mentioned, a stoper was equipped with a control valve on the feed-leg. This machine was laid on a board with holes at intervals, into which was inserted an eyebolt to take the feed-leg pressure.

Former practice in this stope was to bar down, and drill and blast on each shift. This cycle was changed to cover a two-day period, giving much more drilling time per shift, with a reduction of cost as noted below.

All mining is done on day shift. There are two shifts per day, with 4 hr. intervening between shifts. Only muckers on development work are on the second (night) shift. The production in tons per man-shift underground is 12.7.

The main haulage drifts are 8 by 10 ft. in cross-section, enlarged slightly at the loading chutes. No timbering is needed. Drilling is done with wet machines, mounted on vertical bars. Two machines and two drillers average 5½ ft. advance per shift. Three men hand-muck this advance on the night shift. Both drillers and muckers are paid contract wages per foot of advance, powder and equipment being furnished by the company. A round generally consists of 24 holes, although the cut frequently requires more holes. A "burned" cut is the only satisfactory one for all rock.

Raises are 5 by 7 ft. Two drillers, using wet stopers, average 5 ft. advance per shift in both ore and rock. The labor is by contract, with materials and powder furnished. To make 5 ft. advance requires 20 holes in rock and 18 holes in ore.

Drills and Drill Steel

Three types of machines are used. Ingersoll-Rand S-49 dry stopers, using unshanked 1-in. hollow hexagon steel are used for stoping. Most of these are equipped with feed-leg control valve. The same machine, with jackhammer front and back head, is used in the sublevel stopes and for blockholing. All jackhammers are dry machines using 1-in. shanked hollow hexagon steel. Gardner Denver Type 339 wet stopers, with unshanked 1-in. hollow hexagon steel, are used for all raises, and type 7KL wet drifters, with 1¼-in. hollow round steel for all drifting and for undercutting the footwall in the thirteenth level stope.

About 11,000 pieces of steel are handled per month in an underground shop on the thirteenth level. Average steel breakage is 2.33 per cent Consumption of steel is 0.0493 lb. per ton of ore broken, and 2.24 lb. per foot of advance on development.

Tramming

Tracks are 24-in. gage, of 40-lb. rail. Automatic side-dump cars, of 70 cu. ft. capacity, hold about 5 tons. The improvement in chutes and equipment on the thirteenth level has greatly reduced the tramming cost, but it is still excessive, as compared to the tramming from the sublevel stopes. Six trammers on the thirteenth level tram 40 per cent of the ore hoisted, while two men on the fourteenth level tram 60 per cent, where the ore passes over the grizzlies before loading. (See comparative figures in Table 1.) In addition to these advantages, there is so little spillage under the new system that the levels are always clean.

Milling

On page 59 the flow sheet shows graphically the course of the ore.

Milling is a simple process, as all recovery is made with magnets from dry ore. Grinding is done with one 24 by 48-in. jaw crusher, followed

TABLE 1.—*Operating Data Washington Mine*

COMPARATIVE STOPING AND TRAMMING COSTS, PER CENT
Costs Made Prior to 1929 Taken as 100 Per Cent

	Shrinkage Stoping Prior to 1929	Shrinkage Stoping in 1931	Sublevel Stoping in 1931	Total Stoping in 1931
Labor	30.52	10.57	12.33	11.49
Drill sharpening	5.11	4.25	3.11	3.67
Power for drilling	20.31	11.95	8.22	9.98
Explosives	3.91	12.44	8.66	10.48
Blockholing in stopes and grizzlies	28.57	12.75	3.55	8.00
Miscellaneous	11.58	10.76	8.56	9.62
Total breaking cost	100.00	62.72	44.43	53.24
Tramming and loading at chutes	100.00	30.42	14.04	22.22

	Shrinkage Stopes Prior 1929	Shrinkage Stopes in 1931	Sub Level Stopes 1931	Total, All Stopes 1931
Actual tonnage broken per drill shift[a]	31.3	69.0	92.0	82.5

COMPARATIVE MILLING COST, PER CENT
Cost Prior to 1929 Taken as 100 Per Cent

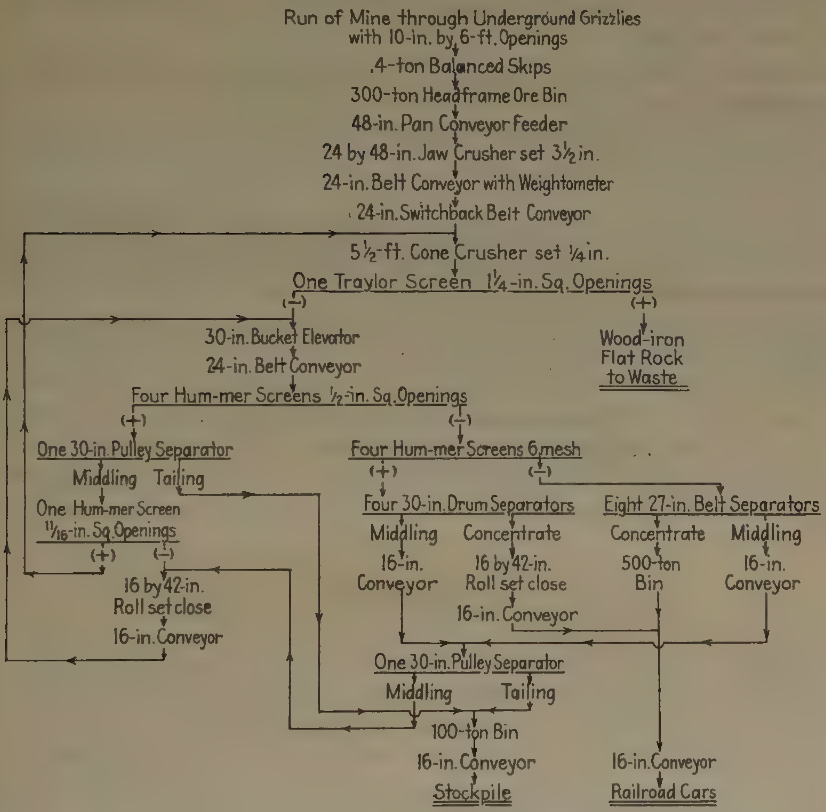
	Prior to 1929	1931
Milling costs	100.00	60.76

AVERAGE ANALYSIS MILL PRODUCTS, PER CENT

	Prior to 1929	1931
Crude	56.37	59.00
Concentrates	60.32	62.94
Total tails	13.11	10.40
Magnetic iron in tails	10.20	6.36

[a] These figures include all shifts charged to stopes whether drilling or not.

by a No. 5 Symons cone, set at 3/8 in.; returns are recrushed by two sets of 16 by 42-in. rolls. Two sets of Hum-mer screens receive the product of the Symons crushers and make two sizes: −1/2 + 1/10 in. treated on four drum separators, and −1/10 in. (6 mesh) on four double-deck belt separators. The tailings from all magnetic machines are further treated on a Dings high-intensity drum, while the middlings, together with the oversize from the ½-in. screen, are reduced by the rolls and recirculated. The tailings go to waste and the concentrates are conveyed directly to 70-ton hopper-bottom cars. The mill capacity is 80 tons per hour, requiring four operators, two loaders and a foreman. All milling is done on one 9-hr. shift and major repairs are made on night shift.


Run of Mine through Underground Grizzlies with 10-in. by 6-ft. Openings
4-ton Balanced Skips
300-ton Headframe Ore Bin
48-in. Pan Conveyor Feeder
24 by 48-in. Jaw Crusher set 3½ in.
24-in. Belt Conveyor with Weightometer
24-in. Switchback Belt Conveyor
5½-ft. Cone Crusher set ¼ in.
One Traylor Screen 1¼-in. Sq. Openings
(−)
(+)
Wood-iron Flat Rock to Waste
30-in. Bucket Elevator
24-in. Belt Conveyor
Four Hum-mer Screens ½-in. Sq. Openings
(+)
(−)
One 30-in. Pulley Separator
Middling
Tailing
Four Hum-mer Screens 6 mesh
(+)
(−)
One Hum-mer Screen 11/16-in. Sq. Openings
(+)
(−)
16 by 42-in. Roll set close
16-in. Conveyor
Four 30-in. Drum Separators
Middling
Concentrate
16-in. Conveyor
16 by 42-in. Roll set close
16-in. Conveyor
Eight 27-in. Belt Separators
Concentrate
Middling
500-ton Bin
16-in. Conveyor
One 30-in. Pulley Separator
Middling
Tailing
100-ton Bin
16-in. Conveyor
Stockpile
16-in. Conveyor
Railroad Cars

FLOW SHEET, WASHINGTON MINE, DRY MAGNETIC SEPARATION.

COSTS

Changes in shrinkage stoping on the thirteenth level have reduced the breaking cost 37 per cent, while the new haulage equipment and improved conditions have reduced tramming costs 69 per cent. Sublevel stoping has shown even better results, notwithstanding the increased cost of stope preparation (Table 1). With sublevel stoping, the breaking cost has been reduced 55 per cent; tramming costs, 86 per cent.

DISCUSSION

(*Lucien Eaton presiding*)

M. E. Johnson,* Trenton, N. J.—Have you made any attempt to do machine mucking at this mine?

C. H. Loux.—No. We do not need to do any mucking, because the broken ore flows by gravity to either a grizzly or the shaft.

M. E. Johnson.—Is not the tonnage mined per day per man-shift higher than any other in the East?

C. H. Loux.—No, I do not think so. Several other operations have about the same figures.

L. Eaton,† New York, N. Y.—I believe that the method described in this paper is more generally known as sublevel stoping than as sublevel caving.

C. F. Jackson,‡ Washington, D.C.—The impression of those unfamiliar with the method is that the cost of sublevel development is excessive Irrespective of the spacing of the sublevels, whether in hard or soft ores the cost per ton is usually about the same, 8 to 10 cents.

L. Eaton.—The cost of development is more dependent on the width of the orebody than on the spacing of the sublevels.

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* Assistant State Geologist of New Jersey.
† Consulting Mining Engineer.
‡ U. S. Bureau of Mines.

Gold Mining in Georgia

By C. S. Anderson, Empire, Colorado

(New York Meeting, February, 1934)

Georgia, since 1829, has produced nearly $18,000,000 from her gold mines, but in late years the output has dwindled to insignificance. In view of present universal efforts to increase gold production, it seems pertinent to record some recent observations in that field, briefly review its past and weigh probabilities of reviving gold mining there. Little in this paper can be claimed as new or original, and it does not present results of a detailed or profound investigation of the whole region.

The Allatoona gold mine was reopened in January, 1932, and explored at greater depth during the period from January to June of that year. A number of other abandoned mines were visited, but since the workings, in common with most such mines in Georgia, were flooded or caved, underground conditions could not be examined. For a wider knowledge of the subject, a search was made through available pertinent literature and statistics, a bibliography of which is appended hereto. To all the sources obligations are gratefully acknowledged.

Production

The discovery of gold in Georgia, in 1829, led to an influx of 6000 to 7000 miners and a production of over $200,000 in the following year. A branch of the United States Mint, established at Dahlonega in 1838, accounted for $6,115,569 up to the time of the Civil War. The output fluctuated greatly from year to year, with an all-time peak of $582,782 in 1843. In only one other year, 1845, was the half million mark exceeded. After the war mining began again, but 1883 is the last year for which an output of $200,000 is reported. Placers, always the main source of production, were yielding less and less, and in spite of increased quartz-mining activities the downward trend continued. The last year in which there was a production of $100,000 was 1902. In 1914, $16,270 came from 26 placers and 11 deep mines, with the latter reporting 1750 tons of $3 average value. In 1925, one placer and four lode mines that mined 8100 tons assaying $1.19 per ton produced $9500. One placer and one quartz mine, operating in 1930, reported a total output of $4194. Table 1 pictures the waning output.

TABLE 1.—*Waning Output of Gold in Georgia*

1830–1883	$14,875,500	1903–1914	$ 589,475
1884–1893	1,127,135	1915–1925	87,490
1894–1902	1,160,457	1926–1930	11,795
		Total 1830-1930	$17,851,852

GEOLOGY AND MINING

The gold-bearing fissure veins and lodes of Georgia occur in several parallel belts of mica or hornblende schists and gneisses of Cambrian or pre-Cambrian age. They are of the lenticular type, mostly of deep-seated origin, and with few exceptions, conform in dip and strike with the foliation of the wall rock. The region was base-leveled in Cretaceous and Tertiary times and the present surface is possibly as much as 15,000 to 20,000 ft. lower than the original one. "Only the roots of once mighty veins are left."

Massive quartz, sometimes bluish, but oftener milk white, is the principal vein filling. The gold is partly free and partly contained in sulfides. Pyrite is most abundant. Chalcopyrite and galena are scarcer and sphalerite is rare. Mica is generally present. Manganese has been noted. Auriferous garnets are reported in one locality. Below the oxidized zone the ore usually has been found to be of too low a grade for profitable operations. Exceptions, where ore has been followed to as much as 500 ft. vertical depth, may be taken as proof of the existence of richer shoots, here and there, in otherwise low-grade veins.

Erosion did not everywhere keep pace with rock decay. Hence the existence of large deposits, often 100 ft. deep, of country rock and quartz decomposed in place, which Becker named saprolites. From these deposits the largest part of Georgia's gold production has been won.

The stream placers first discovered were fairly rich and yielded good returns by hand working. But development of hydraulic mining methods soon made the production from the auriferous saprolites of greater importance. The decomposed material was washed through sluices where the gold from the softer part was saved. The more resistant quartz was subsequently crushed by stamps and its contained gold recovered by amalgamation. Many of these operations were on a large scale; $300,000 is said to have been expended for ditches in one case. That all these operations were financial successes is seriously disputed. Working costs were low, but much gold is said to have been lost. The average value per cubic yard was seldom if ever known, though one 50,000 cu. yd. cut is definitely reported to have yielded 100,000 dwt. But the mass of less definite data leads to the inference that the average value of the ground mined must have been very low.

Though picked specimens of quartz often assayed several dollars per ton, oftener the quartz was barren and the values were found in the adjacent decomposed schist. Also, masses or nuggets of gold were often found in the veins and stringers where they passed into bedrock. But shafts sunk in such enriched portions never found them extending downward beyond 10 ft.

A few modest fortunes were made by open-cut mining before the Civil War, but generally, the early miners found only limited quantities of $10 to $12 ore by mining the richest parts of veins and stringers in open cuts and shallow shafts. Of the hundreds of shafts sunk since the 1830's, only a score are reported as being deeper than 100 ft. Many small mills were built and operated for short periods with indifferent success. A 500-ton plant built in 1900 apparently never started. Ore of too low a grade for profitable operation is the most plausible reason for abandonments. That selective mining below the oxidized zone produced a few thousand tons of $6 to $7 ore may be inferred from scanty operating and production data.

The Creighton (or Franklin) mine is the only one showing a long period of successful operation. Mining and milling began there in 1892 on a 35-ton scale and continued for about 20 years. Ore, followed to a vertical depth of 500 ft. along a 900-ft. incline, occurred in lenticular shoots averaging 3 ft. wide and 75 ft. long. Values varied from a few dollars to $50 to $60 per ton. The average as mined is not definitely known; 40 to 50 per cent of the values were free. Table concentrates were treated by chlorination. Mining and treatment costs, in 1895, were $2.70 per ton. Evidently values decreased in depth, and this, with increasing costs of deeper mining, presumably caused operations to become unprofitable about 1912–13 and the Creighton mine closed down.

THE ALLATOONA GOLD MINE

In the early part of 1932, this old mine, near Allatoona, Georgia, was unwatered, and the original workings were entered and examined. Subsequently, against advice, the shaft was deepened to 100 ft. and the vein drifted on for 230 ft. at that level. Fig. 1 shows the old and new workings, and indicates the lower limit of intense fracturing and oxidation, pitching on a 30° angle along the strike. Three representative photomicrographs, made in connection with the examination, are presented herewith (Figs. 2 to 4).

Little concerning this property has been published. "Word of mouth" knowledge of old mines is notoriously apt to be inaccurate. Information regarding the Allatoona mine was obtained from a presumably trustworthy source. The discrepancies noted between the following quotations and the facts are recorded as a measure of the credibility of much that passes as knowledge of other old mines in Georgia.

"The first operations, 1835–40, produced $260,000. The vein, 4 to 5 ft. wide, was worked to a depth of 60 ft. Ore averaging $12 per ton was mined. Inability to handle the water—the cause for abandonment." These workings are traceable about 300 ft. along the strike, and only by assumption of much higher than $12 values near the surface can the reputed production be accounted for. A few specimens with much visible gold were found by digging in old open cuts. Where the ore had been milled could not be ascertained.

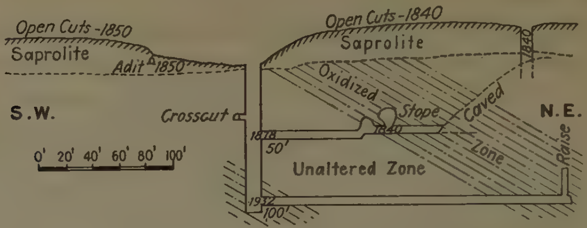
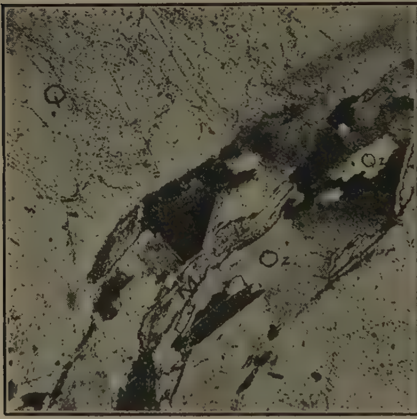


FIG. 1.—LONGITUDINAL SECTION ALONG ALLATOONA VEIN.

"In the 1850's, $5 to $6 ore was mined, from another part of the vein, to a depth of 25 to 30 ft." These workings are traceable over a distance of 600 ft. A small dump of mill tailings, evidently derived from them, assaying $1.03, seems to substantiate the values as reported.



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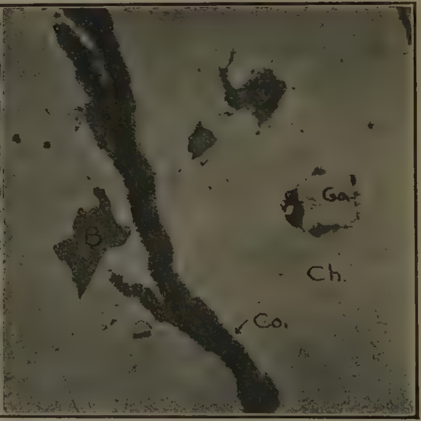

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FIG. 2.—PHOTOMICROGRAPH FROM THIN SECTION. × 50. CROSSED NICOLS.

This shows one of the small fracture zones in the quartz, which has been healed by quartz and mica. *Q* indicates quartz that has been fractured; *Qz*, quartz healing the fracture; *M*, muscovite.

The fine, dusty looking specks in the quartz are small cavities, filled with liquid sometimes showing gas bubbles.

FIG. 3.—PHOTOMICROGRAPH FROM POLISHED SECTION. × 50. REFLECTED LIGHT.

Ga indicates galena; *Ch*, chalcopyrite; *B*, bornite; *Co*, covellite.

This shows the alternation of chalcopyrite to covellite, along a fracture plane in the chalcopyrite. The chalcopyrite and bornite are primary minerals. The covellite is secondary.

"In 1878, an 80-ft. shaft was sunk to reopen the mine. When a short crosscut broke into the original workings the shaft was flooded to the collar. Inability to handle the water again caused abandonment." When unwatered in 1932, this shaft was found to be 65 ft. deep. At the 50-ft. level a 75-ft. drift was found in addition to the short crosscut. Accidental loosening of some lagging in the shaft disclosed another crosscut at 40 ft. At the face of the 50-ft. level a man-sized hole, near the back, led to an old drift. There was no broken rock at the breast. The workings had been pumped out and reentered, presumably immediately after flooding. It seemed more than coincidence that later a foot of ore assaying $12.40 was found in one of the old stopes.

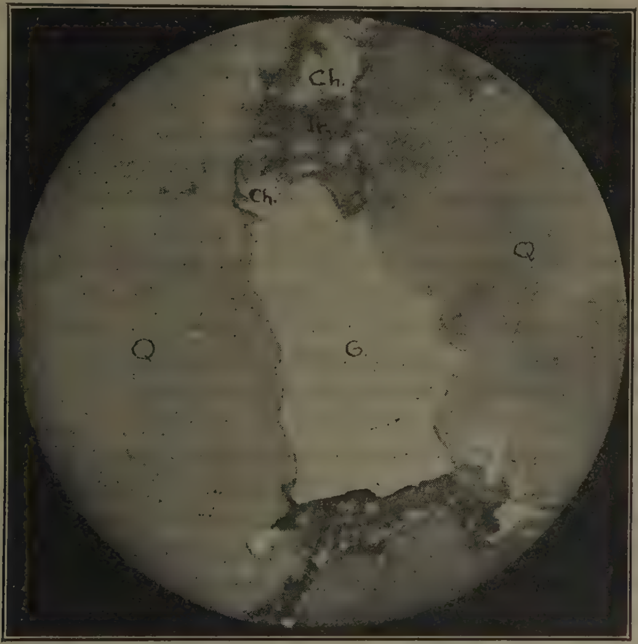

FIG. 4.—PHOTOMICROGRAPH FROM POLISHED SECTION. × 50. REFLECTED LIGHT. *G* indicates gold; *Ch*, chalcopyrite; *Ir*, iron oxide; *Q*, quartz.

This shows the relation of gold to the chalcopyrite. The oxidation of chalcopyrite, resulting in iron oxide, is shown in areas immediately adjacent to gold.

The 75-ft. drift exposed a 2-ft. vein of barren quartz. Wall rock and vein filling showed only slight oxidation. The first stope in the old workings had 5 ft. of barren quartz, but it and the inclosing schist were fractured and stained by limonite. The intensity of fracturing and oxidation increased as the old workings were penetrated. In the second stope, wall rock and quartz were heavily fractured and deeply stained by limonite. The 5-ft. vein was barren, except one foot on the footwall, which assayed $12.40 and showed vugs resulting from leaching of sulfides. Black mineral grains found in fractures and vugs proved to be iron oxides, not manganese, as at first supposed.

The petrographic report showed the gold to be "essentially contemporaneous with the quartz. Two periods of fracturing were shown. One of these, a very minor one, took place during mineralization, and was partly healed with quartz and mica (Fig. 2). The second fracturing came much later, cut across the earlier fractures, and displaced quartz and sulfide grains. Healing of these later fractures has not been well effected."

Enrichment by oxidation of sulfides and perhaps some downward migration of gold together with the easy and cheap working of the softer, weathered portion had made earlier operations profitable. Primary ore probably would be of too low a grade. The poor showing in the old workings gave no encouragement for deeper exploration. Consequently, it was advised that no more work be done.

However, deepening of the shaft and exploration at the 100-ft. level was ordered and carried out. Near its bottom the shaft entered a fractured zone, which the drift on the 100-ft. level passed out of in about 20 ft. Solid, unoxidized rock then continued until a point about 200 ft. from the shaft was reached. The vein, 2 to 3 ft. wide, showed a few specks of gold here and there, but the highest assay obtained was $1.80. Galena, pyrite and chalcopyrite, apparently unaltered, occurred in limited quantity. Concentrates made by panning assayed $4.

Oxidation, beginning about 200 ft. from the shaft, increased in intensity until at 230 ft. the schist was as soft and weathered as on the 50-ft. level. Secondary copper minerals appeared, and at one point complete leaching of sulfides was shown by vuggy quartz. Vertical distance below outcrop was about 125 ft. and below water line about 100 ft. A short raise in the 4-ft. wide barren quartz vein showed a narrow stringer in the hanging wall carrying partly altered sulfides.

A photomicrograph of a polished section of vein rock from this level (Fig. 3) shows the supergene alteration of chalcopyrite to covellite. Fig. 4 from the same specimen, shows the relation of the gold to the sulfides. The petrographic report stated that "the sulfide mineralization is essentially contemporaneous with the quartz, but apparently related to the later stages of vein filling, as evidenced by sulfide minerals cutting across earlier formed quartz grains." No pay ore was found on the 100-ft. level and work was stopped in June, 1932.

Secondary Enrichment

At the Allatoona mine a saprolite deposit overlies a fractured zone in which profitable mining was carried to a shallow depth. Also, indications exist of a similar zone underlying the less profitable workings. Such shattered zones may have been contributing factors in the creation of the saprolites. The subject is of theoretical interest only.

These fractured zones, of course, are particularly favorable for enrichment by oxidation and leaching of the less valuable minerals. Most of the deposits worked in Georgia were probably in such ground. Downward migration of gold probably also occurred. A small amount of manganese contributed by the wall rock would suffice for the solution of some gold, but this gold could not be transported far before being redeposited. The masses of gold found at slight depth below saprolites may owe their existence to such reactions.

It is not implied that the secondary enrichment may not have taken place where no saprolites were formed. For example, about half a mile from the Allatoona, on top of a low hill with only weather-cleaved schist outcrops, a $2000 pocket was found within the last 20 years. The gold all came from an excavation 16 ft. long, 6 ft. wide and 8 ft. deep, and only barren quartz was found below. Specimens of the ore showed friable quartz, with a relatively large quantity of gold consisting of small grains only lightly adhering to each other.

In Georgia the permanent water level is generally a comparatively short distance below the surface. Enrichment below that point is not to be expected.

Conclusions

The placers in Georgia are exhausted. Unexploited areas may exist, but probably are too low in grade and too limited in size for profitable operation.

Prospects of developing another quartz mine comparable to the Creighton are slight. To quote Dr. Graton: "Likely no new mines will be found. Most mines possessing signs of promise have been given a chance." Written in 1906, it seems as true now as then. The possibility of large-scale operations is suggested by some outcrops of wide veins, but consideration of available information makes it extremely doubtful whether any considerable tonnage of workable grade would be shown by a systematic sampling of such croppings.

Had the information presented been of an encouraging nature, this writing would, indeed, have been a pleasant task. It is to be regretted that there seems no reason to hold out hope for a revival of gold mining in Georgia.

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The Red Ore Mines of the Woodward Iron Company at Bessemer, Alabama

By T. C. DeSollar,* Bessemer, Ala.

(New York Meeting, February, 1933)

The group of iron-ore mines known as the Red Ore Mines, owned and operated by the Woodward Iron Co., is on Red Mountain approximately ten miles southwest of Birmingham, Ala. At the present time No. 1 and No. 3 mines are active, No. 2 mines having been abandoned in 1928.

Ore from this group of mines was first mined in 1882, and originally came from open cuts along the outcrop, but, when the "soft ore" (weathered hematite) was mined out and the harder ore was encountered, it became necessary to develop the orebody by means of slopes driven in the ore formation. The three mines to date have produced more than 22,000,000 tons, and the two operating mines now have a capacity of over 1,000,000 tons per year.

Geology

The U. S. Geological Survey has described the features of the district. It places the hematite of Red Mountain, commonly called "red ore," in the Clinton formation of the Silurian system. This formation varies from 200 to 300 ft. in thickness, and is composed largely of shale (slate) and sandstone. There are four iron orebeds in the formation, known as the Hickory Nut, Ida, Big Seam and the Irondale seams. The strike of the orebeds is northeast and southwest, and the dip is southeast, the outcrops being on the west side of the mountain. The two seams that have been worked at the Red Ore mines are the Big Seam and the Ida. The Big Seam, which is and has been the principal source of the ore mined, is approximately 20 ft. thick and consists of hematite and ferruginous bands of sandstone. It is separated into "upper" and "lower bench" ore by a slate parting that is about 30 in. thick. All present mining is confined to the "upper bench" ore, which is uniform in texture and drills and breaks fairly easily when the holes are placed so as to take full advantage of the face of the ore. Typical analyses are shown in Table 1.

The presence of two water-bearing formations, the Fort Payne chert and the Hartselle sandstone, lying respectively 160 and 300 ft. above the Big Seam, make it essential that in mining the ore any movement of ground, which might permit the inflow of water from these formations, should be prevented.

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* Woodward Iron Co.

TABLE 1.—*Typical Analyses of Upper-bench Ore*

	No. 1 Mine, Per Cent	No. 3 Mine, Per Cent
Silica	13.13	13.86
Alumina	3.44	3.54
Lime	17.31	16.03
Magnesia	0.85	0.80
Iron	35.38	36.11
Phosphorus	0.29	0.30
Manganese	0.095	0.10

The orebed has been warped and flexed into folds, accompanied by a number of faults, which have necessitated the use of underground hoists to facilitate the transportation of the broken ore to the underground storage pockets.

NO. 1 MINE

Because of the existence of a number of faults, No. 1 mine is opened by a series of slopes and crosscuts. The mine is bottomed at about 7500 ft. horizontally and 1400 ft. vertically from the mouth of the top slope. This slope is 30 by 10 ft. in cross-section and extends an incline distance of 1880 ft. on a dip of 23°30′ to the first fault. Hoisting is done in a 15-ton self-dumping skip, which is equipped with 24-in. wheels and roller bearings, and is counterbalanced by a car, which travels up and down the slope on a parallel track. The hoist is driven by an 800-hp. motor. The skip is loaded at a measuring box directly under a storage pocket, and discharges into a bin in the tipple above a gyratory crusher. The crushed ore is still further reduced by being passed through a cone crusher, and is loaded into railroad cars for delivery to the blast furnaces at Woodward.

The second slope is 20 by 10 ft. in cross-section and extends 1450 ft. on the dip to meet a fold, which terminates in a fault. In this slope, ore is hoisted in 2-ton cars in trains of five, by a 500-hp. motor-driven hoist. At the bottom of the slope there is a car yard, and at the top there are scales and a rotary car dump, which discharges into the storage pocket of the top slope. From the bottom of the second slope a crosscut was driven 740 ft. in rock to cut the orebody on the opposite side of the fault, and all ore is hauled through this crosscut in trains of 10 cars, by a trolley locomotive.

The third slope was driven from a heading on the far side of the fault, and, after extending approximately 1010 ft. on the incline, flattens to the horizontal and then turns up to cut another major fault. Trains of five cars are handled by a 400-hp. motor-driven hoist in the inclined portion of this slope and by a trolley locomotive in the horizontal part. Beyond the fault at the bottom of the third slope there is an anticline, and in order to reach the ore on the near side a rock-slope was driven upwards through the fault, and was continued over the top of the anticline and down the far side. At the top of the anticline there is a wye-switchback track, at the highest point of which there is a 700-hp. hoist, which hauls trains of ten 2-ton cars up one slope and lowers them down the other without uncoupling. Beyond this anticline the orebed flattens, and trolley locomotive haulage is used.

The normal daily output of this mine, when working two 8-hr. shifts per day, is 2000 to 2200 tons.

No. 3 MINE

No. 3 mine is opened by two slopes and a pilot manway. The mine is bottomed 7700 ft. horizontally and 1445 ft. vertically from the mouth of the top slope. This top slope is on an 18° dip, and extends 3636 ft. incline distance to a fold, which terminates in a fault. The slope was continued in slate approximately 400 ft. to reach the bottom end of a 2300-ton storage pocket, which serves the orebody on the far side of the fault. The top slope is 20 by 10 ft. in cross-section, and hoisting is done in balance with two 10-ton self-dumping skips. The skips are equipped with 24-in. wheels and roller bearings. An 800-hp. motor-driven hoist handles the skips from the skip measuring boxes, located under the storage pocket, to the tipple, where they discharge into a bin above a gyratory crusher. The crushed product passes over a screen-grizzly, where two products are made and loaded separately into railroad cars.

Above the 2300-ton storage pocket on the far side of the fault is the rotary car dump of the second slope. This slope is 2539 ft. deep on the incline; it was stopped by a second major fault. A 700-hp. motor-driven hoist handles all cars on this slope. The trains are made up of either six 2¼-ton wooden cars or three 5-ton steel cars. Diamond drills were used to prospect the fault, and a pilot manway was driven to the ore above, and followed it for several thousand feet. A crosscut 8 by 12 ft. in cross-section has just been driven 2060 ft. to handle the ore beyond the last fault. Loaded cars will be either lowered by gravity to the crosscut yard or hauled to it by a locomotive.

No. 3 mine has an average daily production of 3000 to 3300 tons per day of two 8-hr. shifts.

TRANSPORTATION

Tracks.—The skip-track in No. 1 mine is laid with 90-lb. rails, and has a gage of 60 in. The skip-tracks in No. 3 mine are laid with 80-lb. rails and have a gage of 56½ in. This track is slag-ballasted its entire length.

All other slope-tracks in both mines and all of the tracks in the mechanical loading places are laid with 60-lb. rails. All scramming places and mule-haulage places use track laid with 30-lb. rails. Creosoted ties are used on all slopes and on all permanent heading tracks, and untreated ties are used on all temporary tracks. All electric trolley locomotive tracks have arc-weld bonds.

Cars.—All cars in No. 1 mine are of wood construction, and those in No. 3 mine are of either wood or steel construction, the steel gradually replacing the wood. All cars are equipped with 16-in. cast-iron wheels and roller bearings. The cars are periodically lubricated by an air-driven lubricating machine, which is mounted on wheels and is taken from one mine to the other as needed.

Locomotives.—Trolley locomotives have replaced mule haulage in all mechanical loading places and in all headings beyond the last major fault in both mines. The direct current used to drive these locomotives is supplied by three 200-kw. motor-generator sets located in advantageous places in the two mines.

Mule Haulage.—Mules are still used underground to handle cars from heading and scramming places, where it is not practical to install expensive equipment. The underground mules are housed in a well ventilated, well drained, and well lighted underground barn, furnished with concrete feed boxes, which opens off a 700 by 30-ft. runway.

All hoisting equipment, tracks, trolley and haulage equipment are put in and maintained in accordance with standard practice.

Old and New Methods of Mining

When slope-mining first started, it was customary to turn off headings, or drifts, every 60 to 65 ft. along the slope. These headings were driven 25 to 30 ft. wide, and upsets to the headings above were driven at regular intervals to facilitate ventilation. Drilling was done with piston drills mounted on tripods, and, as the headings were driven full width on the advance, development was not rapid.

For the past 10 years or more, owing to the mechanization of the mines, headings usually have been driven 18 ft. wide, but later practice has been to reduce this width to 9 ft. when the ore is loaded by hand and to 12 ft. when it is loaded mechanically. The narrow headings are driven rapidly to the property line, and in front of the active stopes are widened enough for a double track. The scraper ramp, or slide, used for loading in the headings, is mounted on wheels, and has a 25-hp. double-drum hoist set in front of and above the inclined ramp. The drag-scraper weighs 400 lb., and is handled by two ⅝-in. wire ropes. All drilling in the headings is done by drills mounted on columns and arms. A heading crew consists of a drillman and his helper, a drag engineer, a drag mucker and a motorman. The drill crew works on the shift opposite to drag crew.

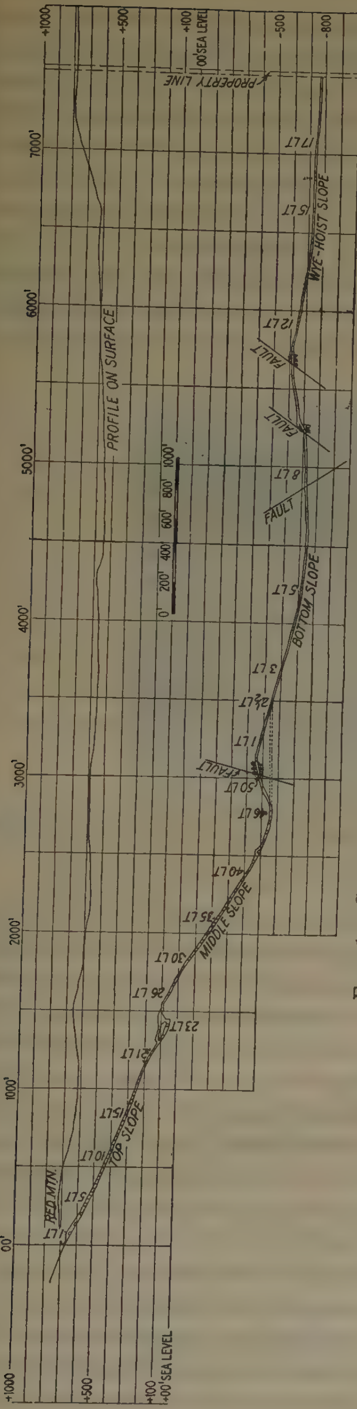

Fig. 1.—Section through the slope of No. 1 mine.

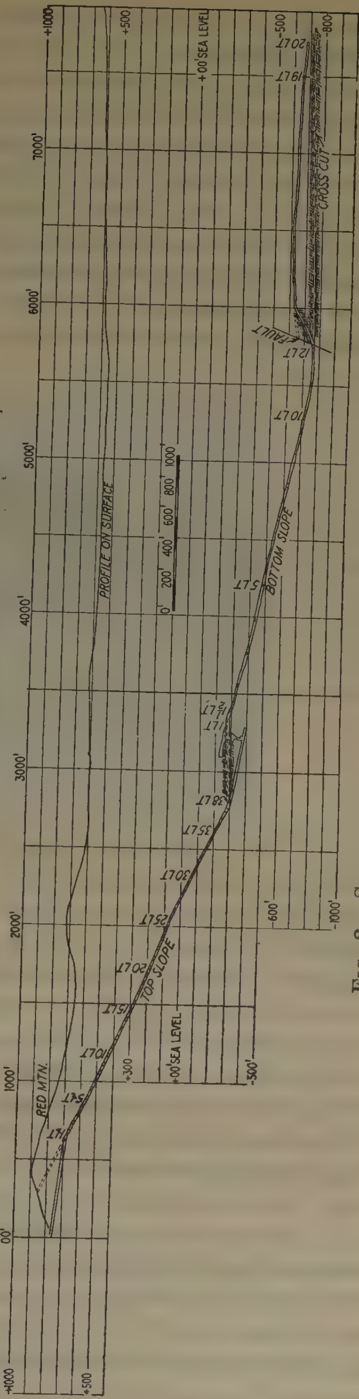

Fig. 2.—Section through the slope of No. 3 mine.

Beginning 25 ft. from the property line and retreating from the face, stope necks are driven up the dip at 75-ft. intervals. Each stope neck is 12 ft. wide at the bottom and is driven up for a distance of 25 ft. Loading is done by hand. The all-steel ramp frame is set up in front of the stope in the stope neck, and the ramp floor plates are set in between the uprights of the frame. These floor plates consist of six pairs of 12-in. 25-lb. channels, 16 ft. long, the ends of which have been bent down to the edge of the flanges to form a sort of wedge. These channels are laid with the flanges down, and are held in place by angle irons riveted to their undersides in the proper position to engage with the ramp frame. The sides are built with ramp floor plates also, each side consisting of a pair of floor plates set on edge with the underside angle iron held against the side of the upright. The rear ends of the side plates are flared into the sides of the stope neck. The ramp floor plates extend 12 in. over the track rail nearest to the stope, and their slope is slightly upgrade toward the track. The upright frame is held in place by two rails fastened to the footwall by hanger wedges. The construction of the ramp and the arrangement of the drag hoist, car, drag and ramp are shown in Fig. 3.

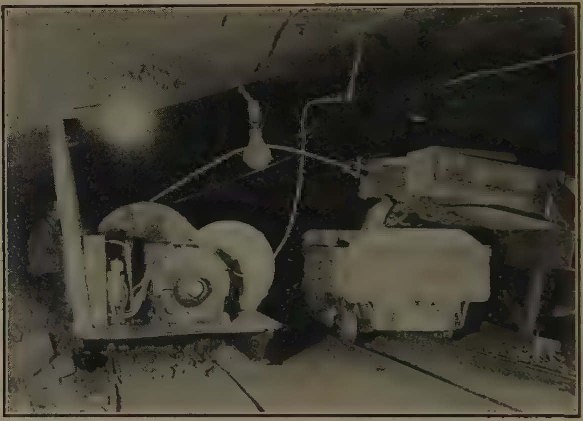

Fig. 3.—Construction of ramp and arrangement of drag hoist, car, drag and ramp.

The top of the stope neck is then widened to 30 ft. and the ore is dragged over the ramp into the car by means of a large double-drum 55-hp. hoist, mounted on a car, pulling a 3100-lb. drag scraper of the box type. The pull-in rope consists of a length of discarded slope hoisting cable, and the pull-back rope is a ⅞-in. plow-steel cable. The stope is driven up 30 ft. wide, or less if necessary, to the heading above, and is then stripped down 10 ft. on each side, making the completed stope 50 ft. wide and leaving a 25-ft. pillar between stopes. From three to five stopes are served by one drag engine, which travels under its own power on a track parallel to the ore-car track in front of the loading ramp. Two stopes are filled with as much broken ore as can be blasted on the footwall, one stope then being dragged, while a third is being drilled and blasted. A fourth and a fifth stope are in process of preparation for the loading ramp, so that the ramp may be set up as soon as one of the other stopes is exhausted. The use of two to three drill crews of two men each permits the accumulation of broken ore in several stopes, and allows the drag engine to serve one stope for several days before being moved to another stope. The drag scraper is never removed from a stope until that stope is worked out.

The ore drills and breaks readily. It is drilled from tripods, which permit greater flexibility in the stopes than columns and arms. One drill crew will drill 14 to 18 holes to a depth of 8 ft. in a shift, and the blast will break from 100 to 125 tons of ore. A complete stope crew consists of two to three drill crews, a drag engineer, a motorman, and three drag muckers. The average length of a stope is 220 ft., but stopes 300 ft. long have been dragged successfully. After the stopes have been exhausted, a second slab is taken from the heading; the track, air pipes and water pipes are removed, and the heading is used to store slate from roof falls. The estimated recovery of first mining is 65 per cent, and the horizontal splitting of pillars by means of switchback openings will recover probably 10 per cent more. The pillars left are sufficient to support the hanging wall.

Drilling and Blasting

Hollow domestic drill steel 1¼-in. round is used in lengths of 4, 6, 8 and 10 ft., with star bits of 2, 2⅛, 2¼ and 2⅜-in. gage, respectively. The drill steel is sharpened in an underground blacksmith shop, using an oil-fired furnace for heating, and is tempered in an automatic pyrometer-controlled tempering machine. All repairs to drilling machines are made in an underground shop. Water for the drills and for drinking is obtained from an underground source.

Fresh dynamite is received daily by truck from the manufacturers, and is taken directly to the man car and thence underground to the magazine. The dynamite used is a low-density ammonia powder, made up in sticks 5¾ by 1½ in., and is detonated by No. 6 tetryl caps. Battery shooting is not feasible, on account of the structure of the roof.

Lights and Telephones

All important places in the mines, such as tunnels, slopes, intersections of headings and manways, and at underground hoists, pumps and drag engines, are electrically lighted.

Telephone lines are arranged to provide communication between all central points on surface and underground.

PUMPING, COMPRESSED AIR AND ELECTRIC POWER

All water pumped to the surface comes from two main pumping stations, one in No. 1 mine and the other in No. 2 mine. All underground water is relayed to one or the other of these two pump stations. The pumps in No. 1 shaft are electrically operated; they consist of three centrifugals and one triplex pump. The pump in No. 2 mine is an electrically driven, duplex pump. In No. 3 mine the pumps are all electrically driven; there are two centrifugal pumps and one triplex pump. There are a number of small air-driven pumps, which are used in temporary places. An average of a little less than 1000 gal. per minute is pumped to the surface annually.

Compressed air for both mines is furnished at a pressure of 100 lb. per sq. in. by three electrically driven air-compressors each with a capacity of 2600 cu. ft. of free air per minute. The main air lines are of 12-in. and 6-in. pipe, and there are two air receivers at each mine.

Electric power for the mines comes from the power plant at Woodward, Ala., at 13,200 volts, and is stepped down to 3300 volts at a substation, where it is divided into four lines, which serve the outside hoists at No. 1, No. 2 and No. 3 mines, the compressor plant at No. 3 mine, and the inside of the mine.

SAFETY METHODS AND FIRST-AID ORGANIZATION

Accident prevention is regarded as a major operation by the Woodward Iron Co., and to that end in the practice followed at the Red Ore Mines safety is ranked first in importance, then quality, cost and tonnage in the order given. There are no slogans or prizes used as inducements in carrying out the safety program, but the responsibility of the company and of the individual is stressed. Safety-correction slips are carried by every foreman, and a report as to whether or not any of them have been issued is turned in at the end of every operating shift. The men who get these correction slips are very closely followed up, and those that are careless are soon eliminated. Safety discipline is very rigid. The record for five years is given in Table 2.

TABLE 2.—*Safety Record for the Past Five Years*

Year	Lost Time Accidents	Number of Mines Operated
1928	124	2
1929	153	2
1930	30	2
1931	25	2
1932	11	1 (To Nov. 15)

Protective clothing, such as goggles and hard-toed shoes, must be worn where hazards are likely to cause an accident. Safety rules and practices have been introduced and made standard, and it is intended that every man should know what is required of him, so that he may follow the best practice for safety and efficiency.

Cleanliness as a means to promote safety is rigidly insisted upon both on surface and underground. Every man working at the two mines has received first-aid training and holds a certificate from the U. S. Bureau of Mines. A regular monthly safety meeting is held with an average attendance of 300 to 400 men; all accidents or bad practices are thoroughly discussed and instruction is given in general safety and efficiency.

Centralization of Ore Delivery from Mines of Compañia de Real del Monte y Pachuca

By H. I. Altshuler,* Pachuca, Mexico

(New York Meeting, February, 1934)

The mines operated by the Compañía de Real del Monte y Pachuca, Pachuca, Mexico, are in two districts, the Pachuca, and Real del Monte. The principal area of mineralization is within a rectangle roughly 10 miles from east to west and 6 miles from north to south. It contains a number of veins from 2 to 20 ft. wide, which are exploited by this mining company. The scattered position of the producing mines led to the construction of two mills between 1906 and 1908; the Loreto mill, in Pachuca, averaged 2600 tons daily, and the Guerrero mill, in Real del Monte, milled 1270 tons daily.

On Feb. 1, 1929, the management decided to enlarge the Loreto mill to take the combined tonnage of both mills at that time. This economic measure required the centralization of ore delivery, which would also afford a material saving in the cost of ore transport. The centralization would give the company the following advantages over the former system of operation:

1. Beneficiation at a single mill, closer to source of materials and supplies than the abandoned Guerrero mill. The refinery is also in the Loreto patio.
2. Ore delivery to surface at one point (San Juan Pachuca shaft) instead of at four.
3. Elimination of aerial tramways.

To show what would be required to effect this centralization, it is necessary to describe the means by which the ore was delivered to the two mills, prior to centralization. The heart of the Real del Monte district is roughly four miles from that of the Pachuca section. The producing mines in these two districts were:

Pachuca district:
  Santa Ana mine
  Camelia group

Real del Monte district:
  La Rica group
  Purisima group
  Santa Margarita mine

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* General Superintendent of Mines, Compañía de Real del Monte y Pachuca.

The first three were tributary to the Loreto mill in Pachuca, the others supplying the Guerrero mill in Real del Monte. The method used to supply these mills is shown diagrammatically in Fig. 1.

To achieve the results desired from centralization, two important underground changes were necessary (see Fig. 2). First, the ore from the Camelia group had to be taken through the San Juan Pachuca shaft, the demolition of the lower Loreto bins being necessary for the mill enlargement; second, the ore from Real del Monte had to be delivered by underground haulage to the San Juan Pachuca shaft. This latter step required: (*a*) a good haulage level from La Rica shaft in Real del Monte to the San Juan Pachuca shaft in Pachuca; (*b*) a haulage level in Real del Monte to deliver the ores to a central point near La Rica shaft; (*c*) bins at this central point, as well as at the San Juan Pachuca shaft; (*d*) hoisting and haulage equipment to assure economical and continuous ore delivery. Since much of the ore handled by the company comes from leased claims, it was of prime importance to arrange suitable bin capacity and control of ore delivery to keep the various ores separate.

Scope of the New Plans

The decision in favor of centralization was contingent on a feasible plan for the delivery of the Real del Monte ores to Pachuca. After consideration of various schemes, it was decided to run a haulage drift to connect the 270-m. level of the San Juan Pachuca mine in the Pachuca district with the 550-m. level of La Rica mine in the Real del Monte district. The 550-m. level at La Rica mine is very close to the bottom of the ore horizon in Real del Monte, a fact favorable to the choice of this level for the main haulage, since most of the Real del Monte ore could then be delivered to the haulage by gravity. Another advantage in the choice of this level was the fact that considerable drifting had been done to the east of the San Juan Pachuca shaft in both the San Juan Pachuca and the Rosario mines, a large part of which could be utilized for the haulage drift. An exploration drift was already in progress toward Real del Monte from the east end of the Rosario mine, the face of which, on Feb. 1, 1929, was at *E* in Fig. 3. Beginning on that date, the drift was increased in size, as shown in Fig. 5, and another drift of the same size was started from the La Rica end to meet it. Simultaneously, two lateral drifts were run, one in the old San Juan Pachuca mine and another in the Rosario mine, to connect with the east work in the latter mine, since the old drifts in these mines were either too tortuous or in too heavy ground to be used for the main haulage tunnel. While the main drift was being driven, these lateral drives connected, and the by-pass drifts for locomotive switching were also driven on both the San Juan Pachuca and La Rica ends.

The plan for delivering the Real del Monte ores to the main haulage level was not so involved as the haulage level itself. Since each mine had its own secondary haulage system, it was only necessary to find means for tying these together near La Rica mine. The main La Rica ore pass, which discharged into pockets on the 600-m. level of La Rica mine was utilized for La Rica ore. A plug was put in this ore pass a few meters

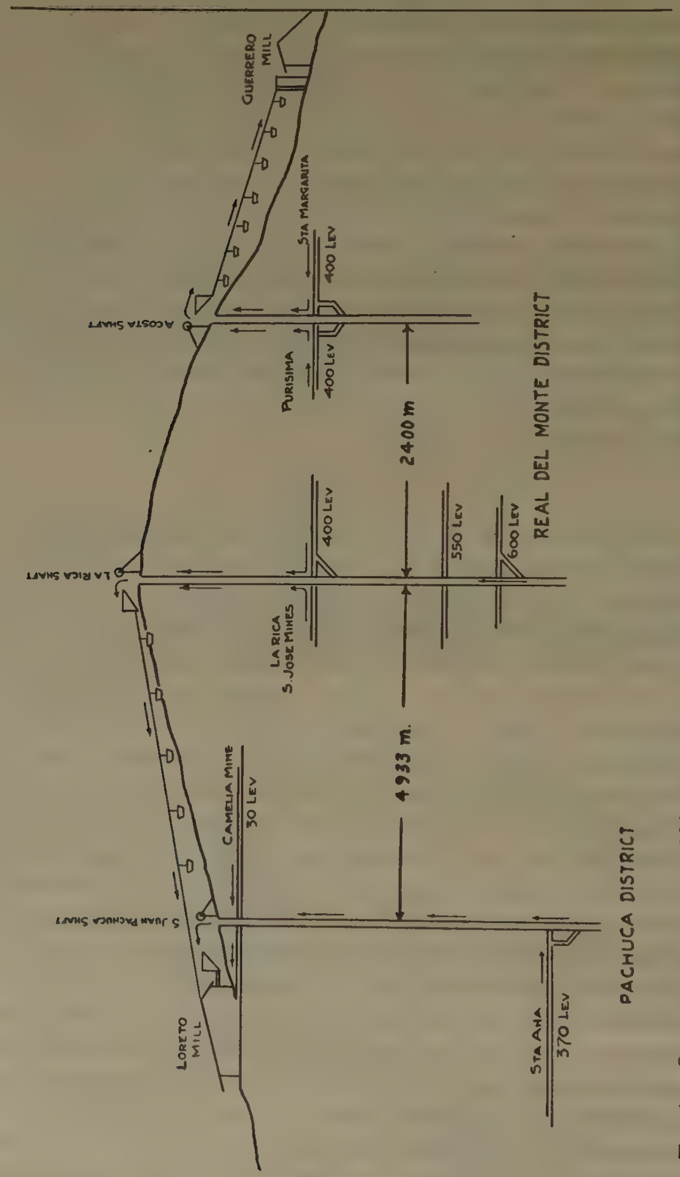


Fig. 1.—Ore delivery, as of 1929 and previous. Not to scale. Arrows indicate direction of movement.

above the 550-m. level, and a short raise was driven to connect the ore pass with the haulage level. The San José ore had been similarly delivered to the La Rica shaft pockets on the 600-m. level. Delivery

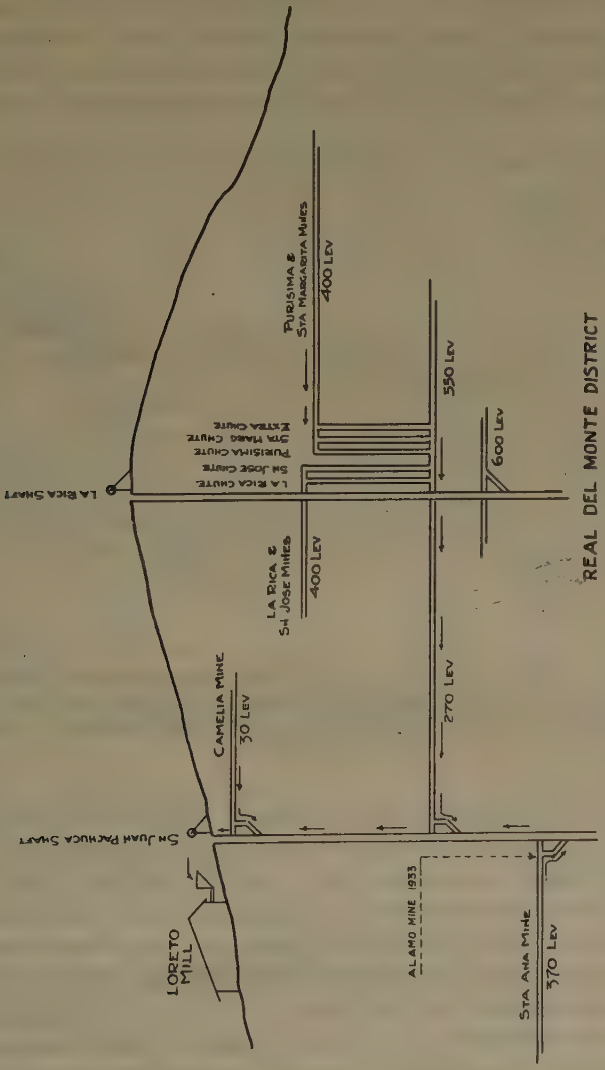


Fig. 2.—Ore delivery, as accomplished by new scheme. Not to scale. Arrows indicate direction of movement.

of this ore to the haulage level was effected in the same manner as that for La Rica ore.

Centralized haulage was more difficult to provide for handling the Santa Margarita and Purisima ores. This was done by driving a new drift from the Hermoso shaft to La Rica on the 400-m. level, through which to transport these ores. Bins for tipple discharge were planned to connect the main haulage on the 550-m. level of La Rica to this new drift on the 400-m. level. It was started from both La Rica and Hermoso shaft sides, and connected Aug. 13, 1929.

As three bins serving the main haulage level were in progress at the time, it was planned to install one new tipple at the end of this new drift for the first bin, and to drive turnouts to the side of the new 400-m. level for the other two dumps. Excavation for the tipples was started as soon as the drifting of the turnouts was completed. One new tipple, made in the shops, was installed after the excavation for it had been made, so that the Purisima ore was handled at La Rica one week before all ore ceased going to the Guerrero mill. During that week the old Purisima tipple was moved from Acosta and set up at La Rica. The Santa Margarita ore was then handled through this tipple, and all ore delivery to Guerrero was stopped Feb. 26, 1930.

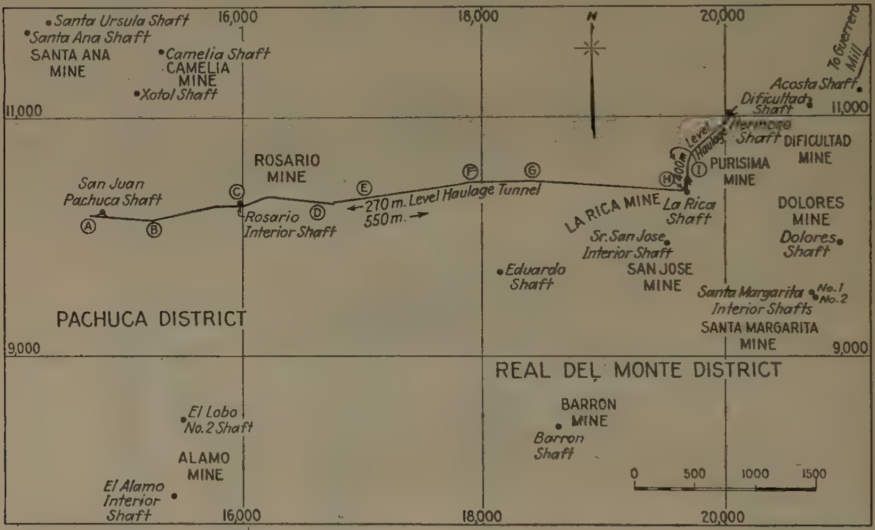


FIG. 3.—LOCATION OF PRINCIPAL PRODUCING MINES OF CIA. DE REAL DEL MONTE Y PACHUCA, SHOWING MAIN HAULAGE TUNNEL, REAL DEL MONTE TO PACHUCA. Scale in Meters

Since the centralization of the mills required reconstruction of the lower mill yard and demolition of the old Camelia bins, Camelia ore had to be hoisted from the San Juan Pachuca shaft in the early stages of the work. The Girault tunnel, through which the Camelia ore had been passing direct to the mill, was quickly connected to the San Juan Pachuca shaft. Delivery of the Camelia ore through the shaft required two bins and a loading chamber, which were started at once. This work was completed July 7, 1929.

TABLE 1.—*Progress of Work for Centralization*

	1929											1930	
	Feb.	Mar.	Apr.	May	June	Jul.	Aug.	Sept.	Oct.	Nov.	Dec.	Jan.	Feb.
Main drive west from La Rica shaft	X	X	X	X	X	X	X	X	X				
Main drive east from E end of Rosario mine	X	X	X	X	X	X	X	X	X				
400-m. level Real del Monte drift to La Rica	X	X	X	X	X	X	X	X					
La Rica lateral driving			X	X	X	X							
Rosario mine lateral driving			X	X	X	X	X	X	X				
San Juan Pachuca lateral driving			X	X	X	X	X	X	X	X			
Widening old drifts and passing switches		X	X	X	X		X		X	X	X		
Excavation for hoisting pockets San Juan Pachuca shaft 30-m. level	X	X	X	X	X	X							
Equipment of hoisting pockets San Juan Pachuca shaft 30-m. level					X								
Excavation for hoisting pockets San Juan Pachuca shaft 330-m. level			X	X	X	X	X	X	X	X	X	X	X
Equipment of hoisting pockets San Juan Pachuca shaft 330-m. level							X		X	X	X	X	X
Equipment of hoisting pockets San Juan Pachuca shaft 410-m. level											X		
Trimming San Juan Pachuca shaft and new steel installation							X	X	X				
Excavation of storage pockets at La Rica	X	X	X	X	X	X	X	X	X	X			
Installation of tipples at top and Gastineau-type gates below									X	X	X	X	X
Installation of permanent track, trolley, and feeder in main tunnel										X	X	X	X
Ore hoist alterations at San Juan Pachuca shaft											X		
Headframe for same											X		
Installation automatic motor-generator set for haulage												X	
Erection locomotives and cars											X	X	
Excavation ore hoisting pockets Rosario interior shaft											X	X	X
Retimbering portions Rosario interior shaft and deepening shaft							X	X	X	X	X		
Pipe lines for water delivery Real del Monte to Pachuca						X	X	X	X	X	X	X	X

The delivery of the Real del Monte ores to the San Juan Pachuca shaft required an elaborate bin arrangement, since it was necessary to keep the four Real del Monte ores separate throughout the haulage and hoisting. Work was started for these bins as soon as a satisfactory plan had been made and the preliminary rock excavation had demonstrated its feasibility.

The ore hoist at the San Juan Pachuca shaft was remodeled between Dec. 25, 1929, and Jan. 2, 1930. The locomotives and cars for the Real del Monte haulage were lowered unassembled and put together underground in the last weeks of the job. On March 1, 1930, the system was put into active operation, delivering all ore from both districts through the San Juan Pachuca shaft.

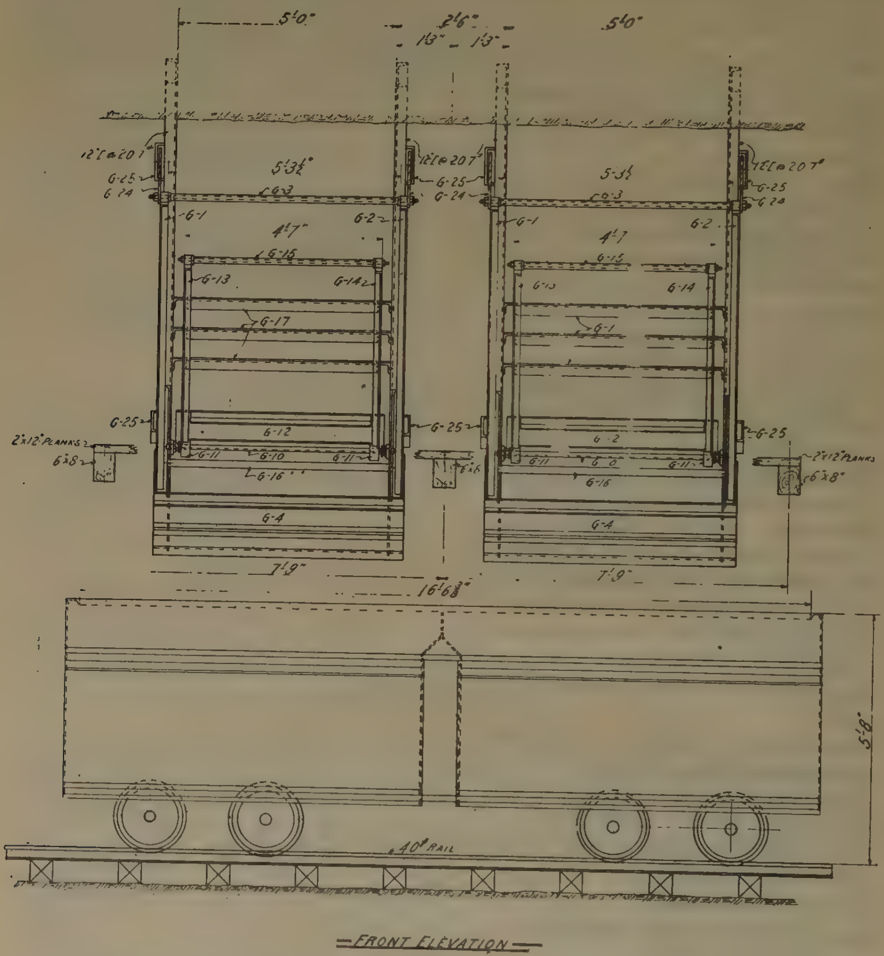
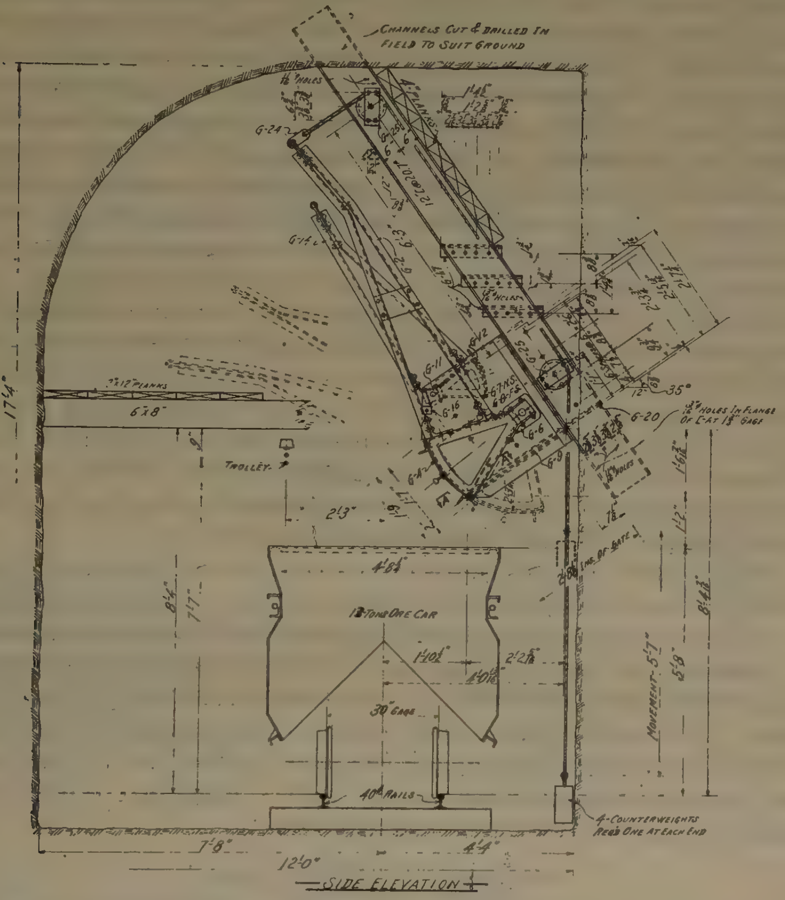


Fig. 4.—Gastineau-type gates

Table 1 shows the order in which the various details of the work were carried on. The Rosario work there mentioned is an auxiliary ore delivery arrangement, which may be used in case of emergency for delivering Real del Monte ore to the Loreto mill. The water delivery system is the arrangement whereby water from the Real del Monte mines is carried through the new workings and pumped to the Loreto mill from the 270-m. level at San Juan Pachuca, in order to give the mill sufficient water in the dry season.



FOR MAIN LEVEL HAULAGE.

Work on the 400-meter Level in Real del Monte

The 400-m. level is connected to all mines in the Real del Monte district; however, to provide suitable haulage conditions, a new drift 800-m. long was driven to connect the old system to the new bins at La Rica. The drift in this section (Fig. 3) was driven from both Purisima and La Rica ends, waste from it being emptied into old stopes below the 400-m. level. The drift section is 6½ by 7¼ ft., the grade being −0.5 per cent toward La Rica. Two machine drills mounted on a crossbar were used in each face, and the work was continuous for three shifts per day.

The three ore bins, which were connected with the 400-m. level, were started at the same time. They were driven simultaneously from the 450, 500, and 550-m. levels of La Rica mine, and were carried up as short shrinkage stopes on the vein. Short raises were made from the tops of these stopes into the hanging wall of the vein to connect with the new level. The large section of these bins was 12 m. long by 2 m. wide, each bin holding approximately 5000 tons. Turnouts driven over these bins on the 400-m. level allow a train to dump in any bin while a second train is passing. At the top of each bin is a tipple for dumping the 3½-ton cars used on the 400-m. haulage level; these cars are 10 ft. long and only one car is dumped at a time. The excavations for the tipples and foundations were made after the drifting and raising had been completed. A bulldozing chamber is provided under each tipple. The grizzlies are on a 32° inclination, and are made of 80-lb. rail resting on 12-in. I-beams at top and bottom, the flanges of the rails being faced up. It was found desirable to bolt 3-ft. liners of 1-in. steel plate on the flanges of these rails at the point of greatest impact. The spacing of the grizzlies is 10 in. at the top and 10½ in. at the bottom, forming a slight fan effect. A drop-off of 4 ft. at the toe of each grizzly prevents oversize boulders from piling up on the grizzly. A short horizontal grizzly in each bulldozing chamber leads the broken boulders into the bin.

The bottom of each bin has two Gastineau gates for loading the 12-ton cars in the main tunnel (shown in detail in Fig. 4). Similar gates are also installed at the bottom of both ore passes from La Rica mine proper, for handling La Rica and San Jose ores respectively. There are five bins, discharging on the 550-m. or main tunnel level, with 10 Gastineau gates. The Gastineau gates (Fig. 4) have small auxiliary gates, which are closed only after finishing the loading of a train at any particular gate, thus minimizing spillage on the track. They have proved very satisfactory, giving better results than baffles.

Work in the Main Tunnel

The section decided upon for the main tunnel is shown in Fig. 5. It was driven from both Pachuca and Real del Monte sides. The waste from the Pachuca end was hoisted to the surface through the Rosario countershaft; that from the Real del Monte end through La Rica shaft. All drilling was done by three S-70, Ingersoll-Rand drifters on a crossbar. Three shifts were worked in each heading, a round being usually completed every shift.

The main drift (*E-F-G-H*, Fig. 3) was driven east from point *E* and west from *H*, connecting at *G*. Simultaneously, two crews drove the lateral drift between *C* and *D*, which did not interfere with the main drift at *E*, as the muck from the latter was hauled to the Rosario interior shaft. This old drift was not utilized as a part of the main tunnel; it was tortuous and in heavy ground, and would have been difficult to enlarge and straighten to conform to the larger tunnel section. The section between *B* and *C* was enlarged at this time to the new tunnel section. The new drift between *A* and *B* was driven after completing the lateral drift between *C* and *D*. This section includes the bypass drift for switching the locomotive at the unloading end. The section *H-I* was driven simultaneously with the drift for the connection, the last 110 m. being of double width, to give the locomotive a passing switch at the loading end. The section *D-E* was enlarged to full tunnel section after the connection was made. Another double-width section was cut for 110 m. at the Rosario interior shaft for a passing switch. The amount of work done on this level, all in medium hard andesite, consisted of:

	Meters
Total new drift	4,022
Total enlarging	1,309
Total bypass enlarging	220
Total on 270-m. level	5,551

Details of New Drift

Costs (U. S. Currency)

	East Drift from San Juan Pachuca, per Meter	West Drift from La Rica, per Meter
Breaking	$22.29	$20.23
Mucking and tramming	6.27	11.26
Timbering	5.14	0.23
Temporary track and pipe	1.82	2.13
Bossing and bonus to men	17.65	14.48
Distributed charges:		
Drill service, hoisting, shops	18.56	15.45
	$71.73	$63.78

Advance

Month, 1929	Pachuca End, Meters	R.d.M. End, Meters	
Feb.-Mar.	311.10	156.60	
Apr.	186.50	149.40	
May	109.60	153.20	(50 m. timber Pachuca side)
June	34.80	151.90	(30 m. timber Pachuca side)
July	162.25	163.90	
Aug.	150.20	165.60	
Sep.	131.95	158.70	
Oct. (Stopped Oct. 22)	98.00	113.10	(Connected Oct. 24)
Total	1184.00	1212.40	
Pounds 40 per cent dynamite per meter	41.6	38.8	
Meters advanced per round	2.09	1.81	

The surveying for this work consisted in establishing a base line at each end of the tunnel, plumbing the shafts, and carrying lines to the faces by repetition surveys, which showed only small differences. The surface tie-in between the shafts was obtained from the company's very accurate triangulation system. The leveling was done by three independent surveys, which were in close agreement. The headings were holed through on Oct. 24, 1929. The closing error of the surveys proved to be 1/7500.

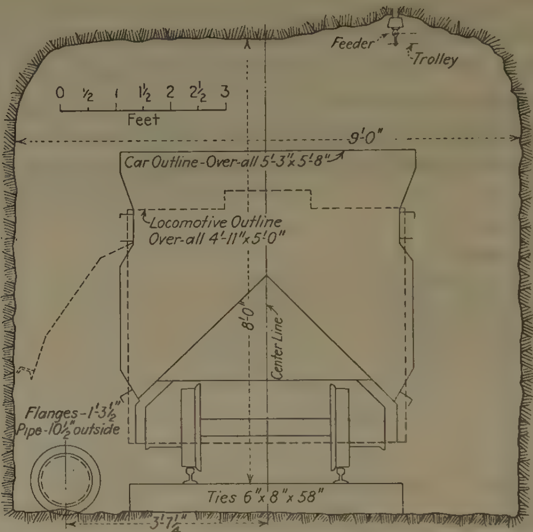


FIG. 5.—IDEAL SECTION, REAL DEL MONTE PACHUCA HAULAGEWAY.

BINS AT SAN JUAN PACHUCA SHAFT

Since, under normal operating conditions, this shaft was to become the only outlet for all ores, bins of large storage capacity were required, so located and constructed that an easy delivery to the skip loading chambers could be effected with the minimum of equipment, dropping the ore the least distance consistent with large bin capacity and ease of operation; this would avoid excessive hoisting depths. The ore was to be delivered on three levels: Camelia ore on the 30-m. level; Santa Ana ore on the 370-m. level; all Real del Monte ore (four kinds) on the 270-m. level.

Below the 30-m. level two bins were required; these started with a pillar of 8 m. between them and came together at the loading chamber 45 m. below the level. As the rock around the top of the east bin was loose, a vertical reinforced concrete collar was put in to a depth of 10 m. The lower portions of the bins were in solid ground, and with careful blasting the work was completed without more concrete than necessary for the I-beams and foundations of the skip measuring equipment. These two bins hold 350 tons each.

The bins on the 370-m. level were not changed. Two bins for ore from the Santa Ana mine, holding 600 tons each, come together 40 m. below the level, the only change required being the substitution of larger skip-loading equipment for the 5-ton skips to be installed. This work was done during December, 1929, without delaying operations at the Santa Ana mine.[1]

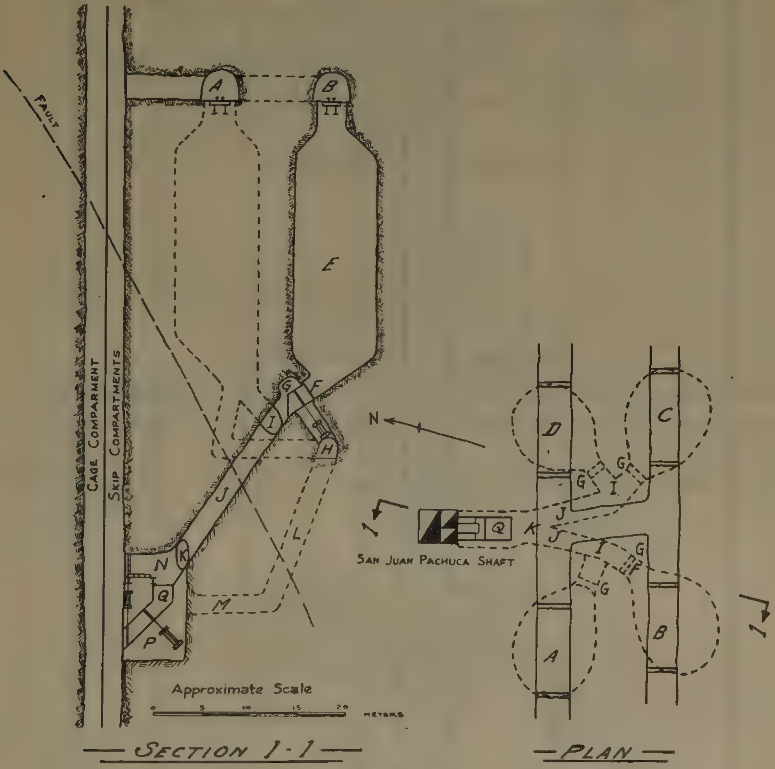


FIG. 6.—PLAN AND SECTION OF BINS ON 270-METER LEVEL.

The bins on the 270-m. level were highly important, since their capacity had to be large and they would have to serve four kinds of ore with dependable and continuous operation. As a south-dipping fault cuts through the San Juan Pachuca shaft a few meters below the 270-m. level (see Fig. 6), the ground on the north side of the shaft was prospected to find suitable ground for locating the bins in the footwall of this fault. Several short crosscuts and a winze showed the rock on the north side

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[1] In 1933, the Alamo mine in the Pachuca district was brought into production. One of the Santa Ana bins on the 370-m. level was utilized for this ore. A raise from the top of this bin was carried to the 270-m. level where the Alamo ore is delivered by a 50-cm. gage haulage system independent of the haulage from Real del Monte (see Fig. 2).

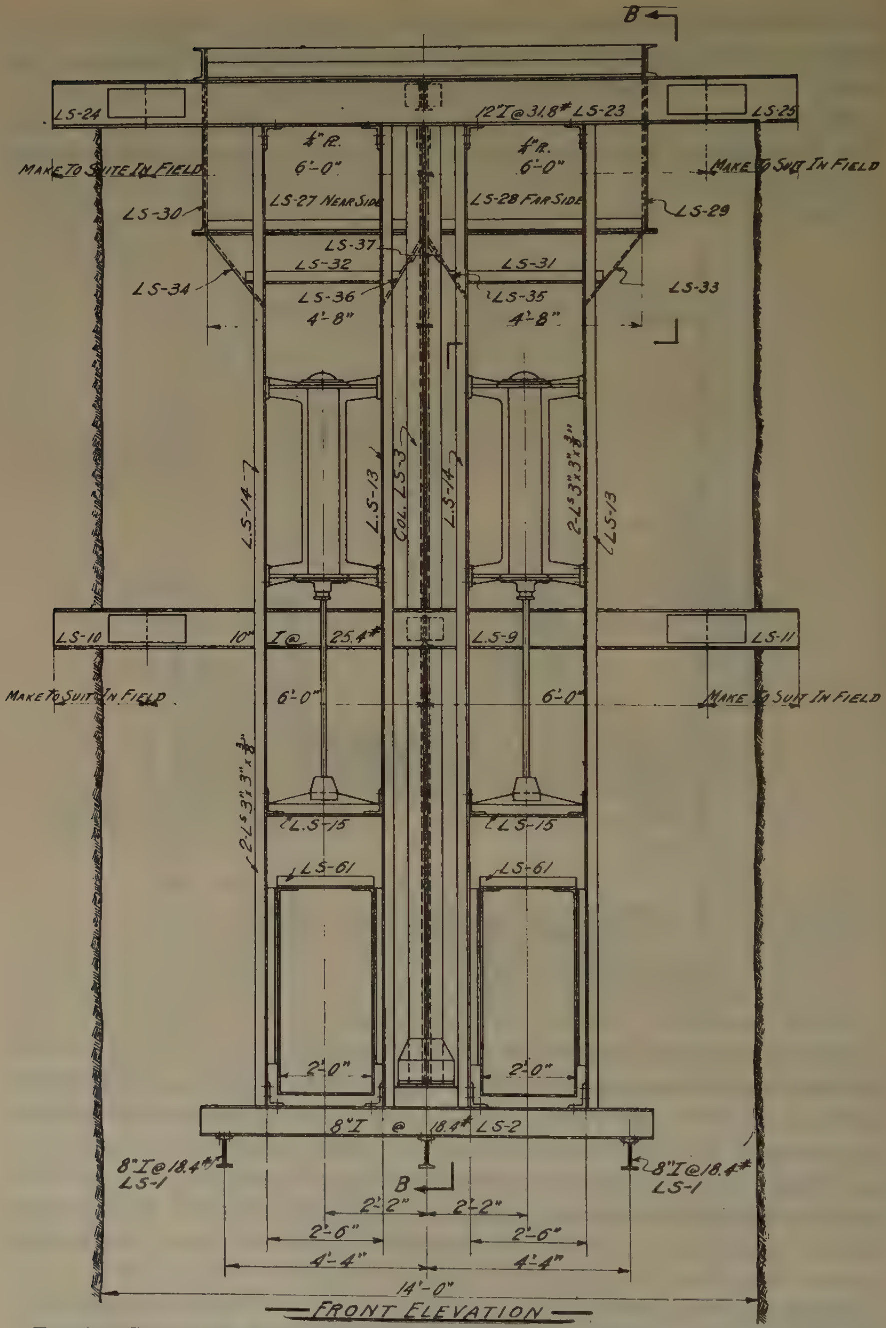

FIG. 7.—SAN JUAN PACHUCA SHAFT, FORTUNA LEVEL. SKIP-LOADING POCKETS.

of the shaft to be too soft and fractured. A crosscut to the south revealed better rock conditions and this side was chosen. The layout was made

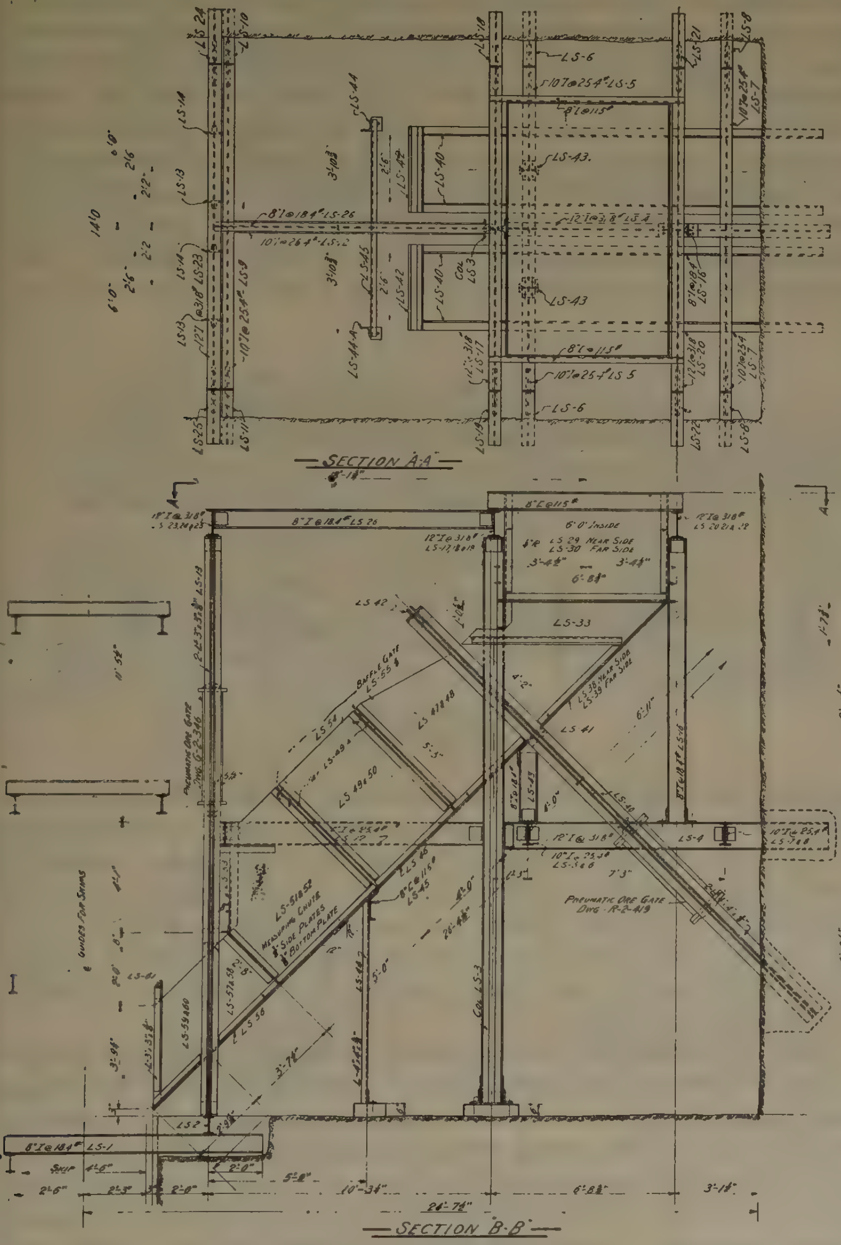

FIG. 7.—(*Continued.*)

so that only bypass raises of relatively small cross-section would cross the fault.

Fig. 6 shows the layout as planned and executed. Four egg-shaped bins (*A*, *B*, *C*, *D*) were so spaced that there was a pillar at least 5 m. thick between them. The main haulage line *B* and the siding *A*, as shown on the section, were projected over the tops of the bins. A trestle, consisting of two 20-in. by 24-ft. I-beams on which were laid the ties and track, was concreted over the top of each bin. These bins were sunk as winzes 2 by 2.5 m., and at the same time the loading chamber *P* was excavated. Two raises, *J*, were started from the back and top of this chamber, *K*, diverging so that at a height of 20 m., at *I*, they had a 5-m. pillar between them. These raises were kept to a 1.7 by 1.70-m. over-all size. At *I*, two small finger raises connected with the bottom of the pockets. The winzes, which were the pilots for the pockets, were enlarged to the full egg-shaped sections, the muck being pulled out through the loading chutes *Q* (Fig. 7) which meantime had been installed. The funneling started 5 m. below the 270-m. level, and, after full size was attained, a series of pieces of drill steel were grouted into holes 1.5 m. apart around the circumference of the bin. Plumb bobs suspended from a point 1 ft. out from the rock gave the drillers a gage to work from. Each bin, except *D*, holds 1800 tons; as bin *D* encountered a bar of soft rock, its section was reduced to one-half.

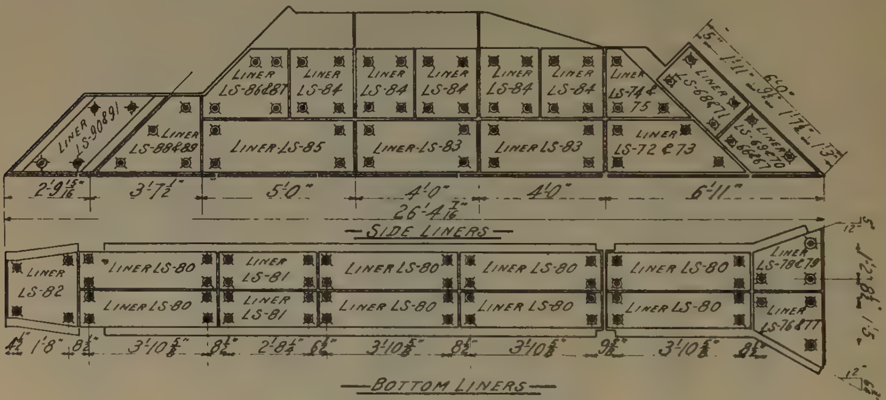

FIG. 7.—(*Continued.*)

After completing the bins and raises, an intricate piece of rock work at *G* made room for the control gates, *F*, for each bin (Fig. 8). These gates, designed to give a 4 by 4-ft. opening, are operated by hydraulic cylinders fed from the pump column in the San Juan Pachuca shaft. As the pillars between raises and bins were small and the hydraulic gate must have a snug fit, the ground had to be cut exactly to the plans, without overbreak. Holes were therefore limited to 2 ft. in depth and were closely spaced by wooden templates. The bottoms of these excavations were connected at *H* by a control drift, semicircular in shape,

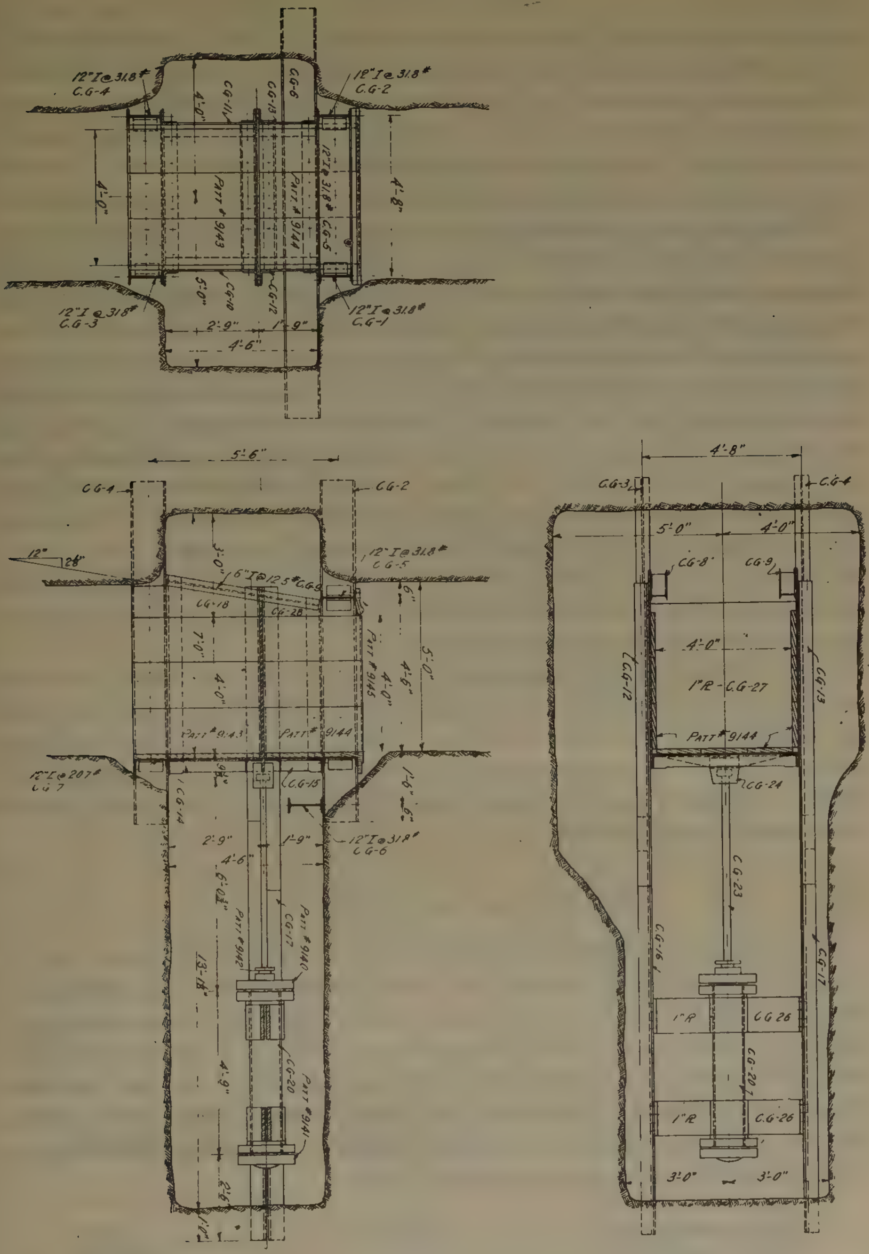

FIG. 8.—HYDRAULICALLY OPERATED CONTROL GATES FOR 330-METER LEVEL ORE BINS.

giving access to each control-gate chamber. This drift, shown on the section (Fig. 6) as *H*, was driven from the top of raise *L*, which started from a short crosscut *M* from the back of loading chamber *P*. The control gates and cylinders (Fig. 8) were lowered through the bins in sections, assembled and grouted in place. The fault previously mentioned, which cut the raises *J*, was tight in the east raise. As in the west it was of questionable strength, a concrete ring was put in where it crossed the raise.

These bins are operated in this manner (see section, Fig. 6): Ore is dumped from the trains into each bin, according to the mine from which it comes, and is held back by the control gates at *G*. As only one kind of ore is hoisted at a time, the gate for the ore chosen is opened and the ore passes through the raises at *I* and *J*, discharging at *K* into the upper hopper *N*. The lower compressed air cylinder is operated, allowing enough ore to run from the steel hopper *Q* to fill the lower channel, which holds one skip load (Fig. 7). The lower cylinder then closes a gate, and as the skip comes to rest under the lower channel, another compressed air cylinder is operated to fill the skip. As hoisting is done in balance, these channels and cylinders are in pairs, so that the opposite one is ready for operation when the downcoming skip arrives. After a sufficient run of one kind of ore the control gate is closed, the raise pulled dry and the system is ready for any other ore chosen.

After putting this arrangement into operation it was found that one ore from Real del Monte, which was particularly sticky, caused trouble in running down the raise. This was corrected by lining the raise with half-round sections of old tube mills and then lining the tube-mill section with half-round manganese steel liners, 15 in. long, 1 in. thick on the sides, and 2 in. on the bottom.

Electric Locomotive Haulage

The haulage equipment for transporting the Real del Monte ores to Pachuca was carefully studied, to insure satisfactory and continuous service. The problem was to handle 1300 dry short tons per 8-hr. shift over a 5-km. haul. At the time of centralization it was thought that two shifts per day, excluding Sundays, would deliver all the Real del Monte ore required. The cars were easily chosen; they were gable-bottom type, of 13 short tons capacity (Fig. 4). Further details of the cars, 18 of which were obtained, are:

Cubic capacity, cu. ft.	380
Weight, empty, lb.	13,200
Weight, loaded, lb.	39,600
Gage, in.	30
Mounting, 8 wheels with outboard axle-boxes and metal bearings	
All body plates, 7 mm. thick	
Bottom plates, 8 mm. thick	

The haul from La Rica bins to the San Juan Pachuca shaft is as follows, all grades in favor of the load: 2808 m. with 0.507 per cent; 924 m. with 0.532 per cent; 863 m. with 0.888 per cent; 338 m. with 0.546 per cent; total, 4933 meters.

LOCOMOTIVES

After an analysis of the proposals of several makers, a Goodman tandem 13-ton unit was selected, as having the greatest margin of overcapacity. The possibility of future increase of tonnage from Real del Monte being recognized, calculations[2] were made for trains of 17 cars and of 22 cars. The former showed that 7 trips per shift (1547 tons) could be expected with 17 cars and 6 trips with 22 cars (1716 tons). Root-mean-square heating calculations (Table 2) showed 136 amp. continuous rating required to haul 17 cars and 164 amp. for 22 cars. The heating curve of the locomotive characteristics was corrected to show a 66° C. temperature rise at sea level, corresponding to a 75° C. rise at 2200 m. altitude. The 164 amp. exceeded the continuous rating by about

TABLE 2.—*Heating Calculation for* 17 *and* 22-*car Trains with Two* 13-*ton Locomotives in Tandem*

Distance, Meters	Grade, Per Cent	Drawbar Pull, Lb.		Speed, Miles per Hr.		Minutes		Amperes		Time × (Amp.)²	
		17 Cars	22 Cars	17 Cars	22 Cars	17 Cars	22 Cars	17 Cars	22 Cars	17 Cars	22 Cars
Loaded train											
2808	−0.507	6,686	8,650	9.8	8.5	11.1	12.8	338	396	1,268,108	2,007,245
924	−0.532	6,518	8,430	9.9	8.5	3.5	4.0	333	390	388,111	608,400
863	−0.888	4,120	5,232	11.5	10.2	2.8	3.2	255	292	182,070	272,845
338	−0.546	6,425	8,312	10.0	8.5	0.9	1.1	320	386	92,139	163,895
Empty train											
338	+0.546	4,590	5,944	10.5	10.0	0.9	0.9	273	317	67,076	90,440
863	+0.888	5,358	6,935	10.2	9.6	3.2	3.4	296	346	280,371	407,034
924	+0.532	4,560	5,900	10.5	10.0	3.2	3.4	272	315	236,745	337,365
2808	+0.507	4,504	5,830	10.5	10.0	10.0	10.5	270	312	729,000	1,022,112
Switching..........						5.0	5.0				
Loading............	level	5,205	6,487			17.0	22.0	254	320	1,096,772	2,252,800
Dumping..........	level	5,205	6,487			8.5	11.0	254	320	548,386	1,126,400
						66.1	77.3			4,888,778	8,288,536

$$\text{Root-mean-square current} = \sqrt{\frac{\Sigma(\text{amp.})^2 \times \text{time}}{\text{total time}}}$$

$$\text{For 17-car train, r.m.s. amp.} = \sqrt{\frac{4{,}888{,}778}{66.1}} = 272 \text{ amp., or 136 amp. per locomotive.}$$

$$\text{For 22-car train, r.m.s. amp.} = \sqrt{\frac{8{,}288{,}536}{77.3}} = 328 \text{ amp., or 164 amp. per locomotive.}$$

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[2] Electric Trolley Locomotives. *Eng. & Min. Jnl.* (1929) **128**.

15 per cent, as shown by the 66° C. heating curve. The continuous rating with blowers is 200 amp. each. The locomotives therefore were ordered with blowers. They have the following specifications:

Voltage	500
Nominal weight, each unit, tons	13
Gage, in	30
Motors mounted tandem with reach gears, 2 motors each unit	
Horsepower each motor	90
Drawbar pull: starting without sand, lb. each unit	7800
running, about, lb. each unit	6500
Speed at running drawbar pull, about, miles per hr	8
Length, each unit, ft	16½
Wheelbase, in	66
Wheel diameter, in	33
Type wheels, steel-tired	
Control—single end contactor; magnetic reverser	

The two units are arranged for separate or tandem operation. The motors are of the heavy railway type, with large ball bearings carrying the armature. The contactor control is advantageous because the controller is only required to carry a small pilot current to operate the contactors on the control panel inside the locomotive frame. Furthermore, in tandem operation the jumper between the two units carries only the small pilot current.

MOTOR-GENERATOR SET

The power input to the locomotives would be 164 amp. each, plus 7.5 per cent for gear losses; total, 176 kw. Provision was made for a small gathering locomotive as a future contingency, 24 kw. being allowed for such service, bringing the total to 200 kw. Addition of 12.5 per cent as a safety factor to cover line losses, etc., led to installing a 225-kw. motor-generator set. Calculation of the peak load on the set was made by using the formulas:[3] $D = 44TV$, and $F = \dfrac{2000(W + L)V^2}{29.92\ D}$. Allowing the train to accelerate in 50 m. (164 ft.) for one locomotive,

$$F = \frac{2000 \times (173.5 + 13) \times (4)^2}{29.92 \times 82} = 2432 \text{ lb.}$$

For a 17-car train, V is taken at 4 miles per hour, and D at 82 ft., since the train is accelerated through half the accelerating distance to half full-load speed with the motors thrown in series. Adding 2432 lb. to the maximum drawbar pull for one locomotive, 3343 lb., the peak drawbar pull is 5775 lb., or 240 amp. per locomotive. Adding 7.5 per cent for gears, plus 35 amp. for a gathering locomotive, the peak requirement on the motor-generator set is 550 amp. Adding 10 per cent for line loss, the total peak load is roughly 600 amp., which is 133 per cent of rated capacity for a 225-kw. set. The period for which this peak is required is figured from the above formula: $T = \frac{D}{44V} = \frac{164}{44 \times 8} = 0.44$ min.

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[3] Where T = time in minutes required to accelerate the train.
F = force in pounds to be added to running tractive effort to obtain total tractive effort while accelerating.
D = distance in feet required to accelerate train.
V = velocity at running speed, in miles per hour.
W and L being the weight of trail load and locomotive respectively, in short tons.

A General Electric motor-generator set was purchased, details of which are:

Construction: Two-unit, 3 bearing induction motor-generator set.
Rating: Continuous 225 kw., 40° C. rise at 8000 ft. altitude; 25 per cent overload for 2 hr. with 55° C. rise; 100 per cent overload permissible for 1 min.
Motor: 2080 volts,[4] 330 hp., 3 phase, 50 cycle, 1000 r.p.m., direct connected to
Generator: 6 pole, 225 kw., 1000 r.p.m., $^{500}/_{550}$ volt, 10 per cent overcompounding. Full load speed 985 r.p.m.
Compounding characteristics:

AMPERES	VOLTS
0	500
300	536
380	547
410	550
570	567

Full automatic control was provided for the motor-generator set. This functions so that, when the starting button is pressed, reduced voltage is applied to the induction motor; this motor is not connected to full line voltage until a definite-time accelerating-relay operates. Meanwhile, the generator builds up voltage and after polarity is checked, the generator is connected to the load by a direct-current recloser. The automatic switching equipment provides protection against:

1. Starting with low a. c. line voltage.
2. Running with low a. c. voltage or with moderately prolonged overload which, if continued, might overheat the motor.
3. Short circuits on the d. c. load.
4. Operation with heating bearings. A thermal relay on each bearing is operated to shut the plant down if any of the bearings get hot.

The direct-current recloser functions as follows:

1. It trips the feeder on any overload beyond the overload setting.
2. It keeps the feeder disconnected for a definite time when tripped by d. c. overcurrent.

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[4] Usual mine voltage is 1040. In this case, as a special bank of transformers on the surface was devoted to the haulage, 2080 volts was chosen in order to reduce the size of the new lead-sheathed cable required.

3. It makes no attempt to reclose as long as an excessive load remains connected.

4. It recloses the feeder when a definite time has elapsed after tripping, provided that the excessive load has been removed and full voltage has been restored.

Trolley, Feeder, and Track

The motor-generator set was placed close to the Rosario shaft, thus splitting the line about one-third the distance from San Juan Pachuca to La Rica. Forty-pound rail was used between San Juan Pachuca and the motor-generator set, and 60-lb. rail for the rest of the line to La Rica. Preliminary calculations showed that a feeder cable would be required from the motor-generator set at Rosario to the La Rica end. After the compounding characteristics of the set were received, a final calculation indicated that a 350,000 circular-mill cable[5] would prevent an excessive voltage drop at any point. The results of calculations for a 17-car train are as follows:

1. Resistances on La Rica side of set:

	Ohm per Mile
60-lb. single rail resistance	0.0877
Add 10 per cent for bonding	0.09647
2 rails in parallel	0.04824
4/0-copper trolley wire	0.264
350,000 c.m. copper feeder	0.163
Parallel resistance trolley and feeder	0.101
Resistance of track plus trolley and feeder	0.14924

2. Voltage drop starting loaded train at La Rica:

Distance from generator about 2.11 miles
Current, 516 amp., 562 volts at generator (compounding characteristics)
Total resistance, 0.14924 × 2.11 = 0.315 ohm
Voltage drop, 516 × 0.315 = 162 volts
Line voltage, 562 − 162 = 400 volts

3. Voltage drop running from La Rica:

Current 363 amp.,[6] 545 volts at generator.
Voltage drop, 363 × 0.315 = 115 volts
Line voltage, 545 − 115 = 430 volts

4. Resistances on San Juan side of set:

	Ohm per Mile
40-lb. single rail resistance	0.1316
Add 10 per cent for bonding	0.1448
2 rails in parallel	0.0724
4/0-copper trolley wire	0.2640
Parallel resistance track and trolley	0.3364

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[5] The Ohio Brass Co. is now manufacturing hard-drawn copper trolley wire in much larger sizes than 4/0. This would be more satisfactory than the use of feeders in many cases.

[6] Current required by motors taken from Table 2, adding 7.5 per cent for gear loss.

5. Voltage drop starting empty train at San Juan Pachuca:
   Distance from generator about 0.947 mile.
   Current, 360 amp., 544 volts at generator.
   Voltage drop, 360 × 0.3364 = 121 volts
   Line voltage 544 − 121 = 423 volts
6. Voltage drop running from San Juan Pachuca:
   Current, 294 amp., 536 volts at generator.
   Voltage drop, 294 × 0.3364 = 99 volts
   Line voltage, 536 − 99 = 437 volts

As the train proceeds from the points calculated toward the motor-generator set the voltage gradually rises to 536 volts, on 300 amperes, which is reached as the train passes the substation.

INSTALLATION AND OPERATION

The track was laid from both ends of the haulage level on rolled steel ties, weighing 55 lb., spaced at 24-in. centers in the 40-lb. rail section and at 26-in. centers for 60-lb. rail. Every third length of track was cross-bonded.

The locomotives and cars were lowered unassembled into the mine at San Juan Pachuca. The first train of ore was hauled through the tunnel on Feb. 10, 1930, and the entire equipment was found in good working condition.

The operation of the haulage system has been satisfactory. A 17-car train was designed to deliver 68,000 short tons per month from Real del Monte to Pachuca without Sunday work. An 18-car train is actually delivering from 90,000 to 95,000 tons. The improved duty is due to the fact that the rolling friction of the cars is only about 15 lb. per ton instead of the assumed 30 lb., permitting a much faster haulage cycle. On the down grade with the loaded train no current is required after the train has been accelerated. On the empty return trip the locomotives are given their full current, which allows this part of the run to be made in 25 per cent faster time than was assumed. A comparison for an average round trip shows:

	Min. Actual Operation	Min. Calculation from Table
Loaded train, downgrade	15	18
Empty train, upgrade	13	17
Loading	17	17
Unloading	8	9
Switching	3	5
	56	66

Actually, eight trips have been made in an 8-hr. shift, though six or seven is the average, due to various operating delays. Haulage is worked three shifts. No haulage is done on Sundays or holidays, which are used to clean up spillage along the track, replace worn rails on curves,[7] and make minor repairs.

Owing to the low frictional resistance per ton, the braking problem has become important. The wear on locomotive wheels, car bumpers and draw-heads has been heavy. Air brakes have been installed for greater safety in braking, as well as to reduce wear. They are placed on six cars spaced at intervals in the train and are operated by a compressor on one of the locomotives. Controls are placed in all locomotive cabs, so that the engineer can use the air-brakes at any time. On the 0.5 per cent downgrade the loaded train can be brought to a full stop from 12 miles per hour in 50 m. with air brakes and hand brakes together; in 125 m. with hand brakes alone. On the 0.9 per cent grade, 65 m. and 170 m. are required, respectively. During the first two years of operation the locomotives were given an inspection and light repair every Sunday. As it was apparent that, with the 35,000 miles travel per year, they would sooner or later need a general overhauling, a third locomotive was purchased in 1932. The blowers ordered with the locomotives are not used, since the power requirements are much lighter than anticipated. There has been no overheating and the locomotives give excellent service.

The motor-generator set is in a locked station and requires no attention except a dusting once a month.

ORE HOISTING

After deciding to centralize the milling in the Loreto mill, it was evident that the San Juan Pachuca shaft[8] would have to be equipped to handle all of the ore for that mill. The problem was to choose between buying complete new hoisting equipment and remodeling an existing hoist. With no Sunday work, it was necessary to figure on a hoisting capacity of 5000 short tons daily, as follows:

18 per cent from the 30-m. level bins; Camelia ore
64 per cent from the 270-m. level bins; Real del Monte ore
18 per cent from the 370-m. level bins; Santa Ana ore

Skips were limited to a capacity of 11,000 lb. each, by the cross-section of the skip compartments and the height of the headframe. The hoist chosen for remodeling was the large Wellman-Seaver-Morgan flat-rope reel at the San Juan Pachuca shaft. It had a 12-in. main shaft and 12-ft. reels operated at 39.5 r.p.m., with a 375-hp. motor, double reduction spur gears, ½ × 5-in. flat rope, and each reel was controlled by a Welch safety device. Realizing the economy of altering this hoist, instead of purchasing a new one, a study of the actual operating conditions revealed the condition shown on the first curves, Fig. 9, which were based on actual operation; and then a set of assumed conditions were calculated to give the curves on the second part of Fig. 9. The basic figures were:

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[7] Curves of 50-m. radius are the sharpest on the line. Rails on these curves wear out in a year's operation. None of the other rails have needed replacement to date.

[8] The San Juan Pachuca shaft is vertical. It has two ore-hoisting compartments 4 × 5 ft. and a cage compartment 5 ft. 8 in. × 13 ft.; 4 × 10-in. guides are used.

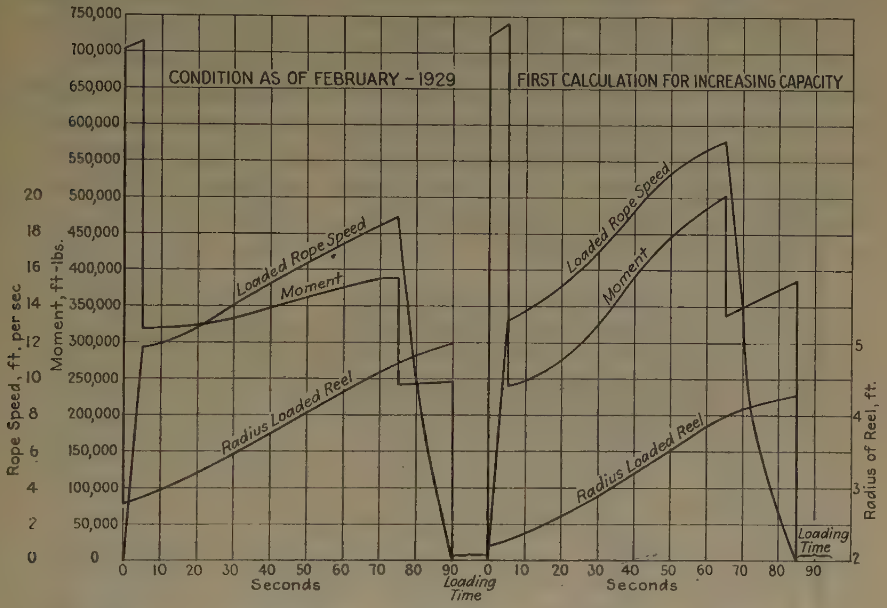

FIG. 9.—HOISTING FROM POCKETS BELOW 370-METER LEVEL, SAN JUAN PACHUCA SHAFT.

	ACTUAL, FEB., 1929	FIRST ASSUMPTION
Weight one skip, empty, lb	5,610	6,100
Weight ore, lb	7,700	10,000
Loading time, sec	10	10
Accelerating time, sec	5	5
Retarding time, sec	15	20
Starting radius, ft	2.75	2.2
Size hoisting rope, in	½ × 5	⅜ × 5
Hoist, r.p.m	39.5	56.2
Trips per hour	36	38
Short tons per hour	139	187
Root-mean-square power at hoist, hp	265	522

In the assumed conditions the starting radius was reduced, making the horsepower smaller than for a larger radius. Other calculations indicated that this hoist might prove adequate. When the problem was submitted to the Wellman-Seaver-Morgan Co., they proposed to increase the speed to 60 r.p.m. and introduce certain mechanical improve-

TABLE 3.—*Hoist Duty Comparisons, Varying Radii and Speeds*

	Calculation No.	Conditions				R.m.s. Hp. at 8000 Ft.	Time Cycle (Seconds)					Trips per Hr.	Short Tons per Hr.
		R.p.m. of Hoist	Minimum Radius at L. Pt. 3, Ft.	Load Point	General		Accel.	Const. Speed	Retard	Load	Total		
A	1	60	2.39	1	Keeping time cycle = cycle used in W.-S.-M. calculations from load point 3. To show hp. variation with varying speeds of hoist	763	10.00	2.09	8.00	15.00	35.09	102.6	282
	2	60	2.39	2			10.00	40.33	8.00	15.00	73.33	49.1	270
	3	60	2.39	3		662	10.00	56.50	8.00	15.00	89.50	40.2	221
	4	52½	3.00	1		688	10.00	2.58	8.00	15.00	35.58	101.2	278
	5	52½	3.00	2			10.00	41.32	8.00	15.00	74.32	48.4	266
	6	52½	3.00	3		647	10.00	56.50	8.00	15.00	89.50	40.2	221
	7	45	3.76	1		622	10.00	2.66	8.00	15.00	35.66	100.9	277
	8	45	3.76	2			10.00	41.68	8.00	15.00	74.68	48.2	265
	9	45	3.76	3		639	10.00	56.50	8.00	15.00	89.50	40.2	221
B	10	52½	3.76	1	Maximum capacities with 750-hp. motor	709	10.00	1.12	8.00	15.00	34.12	105.5	290
	11	52½	3.76	2			10.00	34.68	8.00	15.00	67.68	53.2	293
	12	52½	3.76	3		756	10.00	47.22	8.00	15.00	80.22	44.9	248
C	13	60	4.41	1	Maximum capacity of reel to hold rope	1106	10.00		7.40	15.00	32.40	107.9	297
	14	60	4.41	2			10.00	26.00	8.00	15.00	59.00	61.0	336
	15	60	4.41	3		1030	10.00	35.00	8.00	15.00	68.00	53.0	292

Note: Half-loads (5500 lb.) are used throughout from Load Point 1.
All above operation with reel hoist; ⅜ × 5-in. flat rope; 6100 lb. empty skip.

Tonnages	Hours Operation A			B	C
	60 R.p.m.	52½ R.p.m.	45 R.p.m.	52½ R.p.m.	60 R.p.m.
930 Short tons daily from load point 1	3.30	3.34	3.36	3.20	3.13
3201 Short tons daily from load point 2	11.87	12.03	12.08	10.94	9.54
847 Short tons daily from load point 3	3.83	3.83	3.83	3.33	2.91
4978 Total daily tonnage	19.00	19.20	19.27	17.47	15.58

ments to accomplish the end desired. Their calculations covered one set of conditions for speed and starting radius, namely 60 r.p.m. and 2.35 ft. respectively. Evidently, an infinite number of solutions of the problem could be obtained by varying the hoist speed and starting radius, keeping the duty of the hoist constant. By decreasing the speed, the starting radius would have to be increased for a constant tonnage.

To choose the best solution, the calculations summarized in Table 4 were made. Load points 1, 2, and 3 are the bins at the 30, 270, and 370-m. levels, respectively, representing hoisting distances of 288 ft. from load point 1, 1118 ft. from load point 2, and 1387 ft. from load point 3. Condition A, including calculations 1 to 9, covered three speeds of hoist, 60, 52½, and 45 r.p.m. These all gave approximately the same tonnage in the same hoisting time. Condition B showed the maximum capacity of the hoist at 52½ r.p.m. but limited the motor to 750 hp. Condition C showed the maximum capacity of the hoist at 60 r.p.m. and using the largest minimum radius that the hoist could stand. The column "R.m.s.-hp. at 8000 ft." shows the motor horsepower required, calculated by the root-mean-square method and corrected for altitude by the A.I.E.E. rule.[9] These calculations were based on adopting a single-reduction herringbone drive, giving 80.5 per cent efficiency, WR^2 of hoist 752,000 lb-ft.^{2}, and WR^2 of motor armature of 12,000 lb-ft.^{2} The WR^2 of the inactive rope on reels was calculated for each case.

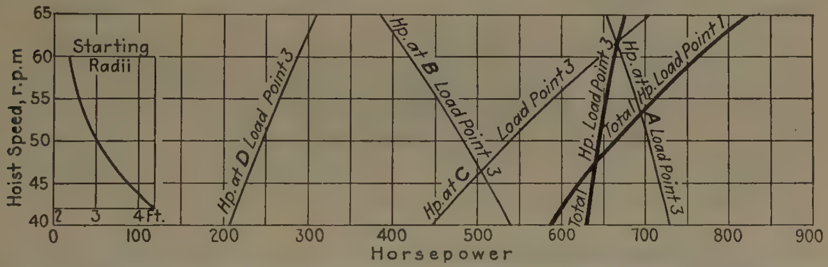

FIG. 10.—HORSEPOWER REQUIREMENTS FOR SIMILAR DUTIES WITH VARYING HOIST SPEEDS. INSERT AT LEFT SHOWS VARIATION OF STARTING RADII.

The curves on Fig. 10 show the details of these calculations. Horsepowers are of interest at the following points:

A = at the end of acceleration the instant before starting constant speed.
B = at the start of constant speed.
C = at the end of constant speed.
D = at the beginning of retardation.

The curves in Fig. 10 show that at 52½ r.p.m. the power required for load points 1 and 3 are nearer to a common requirement than at

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[9] The A.I.E.E. standardization rule No. 308 is a 1 per cent reduction in efficiency for each 100 meters altitude in excess of 1000 meters above sea level.

60 r.p.m. This seemed a more moderate speed for the large hoist, and condition B from the table also indicated that a greater margin of time could be obtained by using this condition. Thus, by decreasing the speed from 60 r.p.m. to 52½ r.p.m. and increasing the minimum radius from 2.39 ft. to 3.76 ft., the same sized motor would be required for doing the work in less than 18 hr. The same work would require over 19 hr. with the faster speed. With this as a final choice, a complete calculation of the horsepower and duty for the three load points was made (Table 4). The results are shown diagrammatically in Fig. 11.

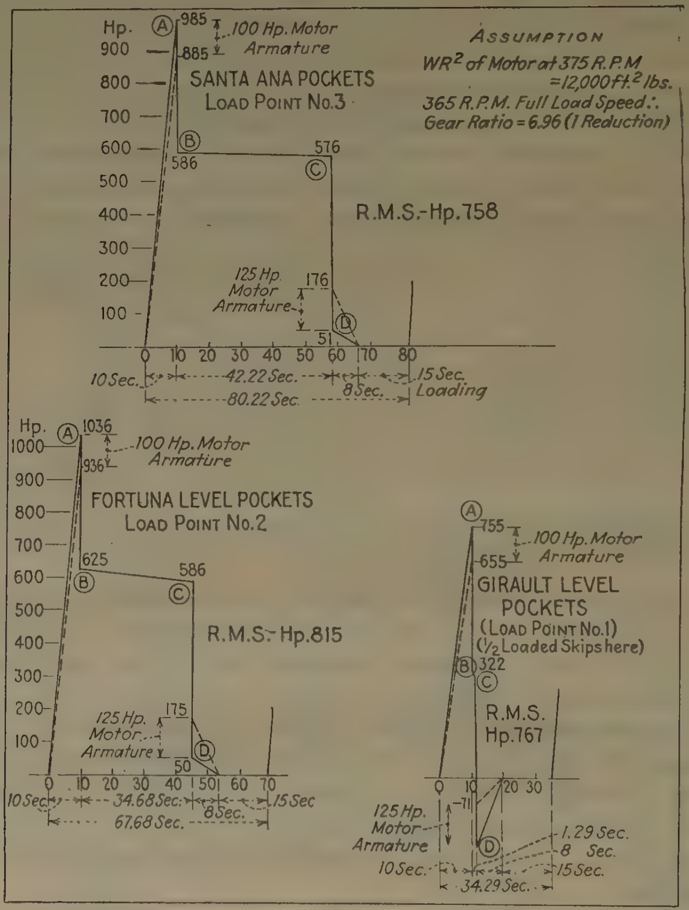

Fig. 11.—Horsepower requirements. Proposed ore hoisting from San Juan Pachuca shaft.

Mechanical Engineering

After choosing the duty cycle it was necessary to study the mechanical and electrical changes required under the new conditions.

TABLE 4.—*Complete Hoist Duty Calculations*

Duty Cycles—Points 1, 2 and 3

	Operation	Time, Sec.	Rate	Total Revolutions	Mean Circum., Ft.	Feet Travel
Point 1 288 Ft.	Acceleration	10.00	0.4375	4.375	31.53	137.94
	Constant speed	1.29	0.875	1.128	32.1	36.20
	Retardation	8.00	0.4735	3.5	32.53	113.85
	Loading	15.00				
	Total	34.29				288.00
Point 2 1,118 Ft.	Acceleration	10.00	0.4375	4.375	26.06	114.00
	Constant speed	34.68	0.875	30.35	29.33	890.15
	Retardation	8.00	0.4375	3.5	32.53	113.85
	Loading	15.00				
	Total	67.68				1,118.00
Point 3 1,387 Ft.	Acceleration	10.00	0.4375	4.375	24.492	107.18
	Constant speed	47.22	0.875	41.33	28.213	1,166.02
	Retardation	8.00	0.4375	3.5	32.514	113.8
	Loading	15.00				
	Total	80.22				1,387.00

Tonnage

	Trips	Skip Content	Tons of 2,000 Lb.	Total Time, Sec.
From Point 1	164	2.75	451	5,625
Point 2	583	5.5	3,206.5	39,467
Point 1	164	2.75	451	5,625
Point 3	163	5.5	896.5	13,076
Total	1,074		5,005.0	63,793 = 17.72 hr.

General Working Data

Speed of hoist, 52.5 r.p.m. or 0.875 r.p.s.

Largest working radius, 5.23 ft.

WR^2 of 750-hp. motor at 375 r.p.m. (365 full load speed) = 12,000 lb-ft.^{2}

Gear ratio required $= \frac{365}{52.5} = 6.96$ (one reduction of herringbone gears)

Inertia $= \frac{12,000}{32.2} \times (6.96)^2 = 18,052$

Angular velocity $= 2\pi \times 0.875 = 5.5$ radians per sec.

Angular acceleration $= 5.5/10 = 0.55$ radians per sec. per sec.

Accelerating moment $= 18,052 \times 0.55 = 9928$ lb-ft.

Horsepower required to accelerate $= \frac{9928 \times 5.5}{550} = 100.$

Horsepower required to retard in 8 sec. $= \frac{100 \times 10}{8} = 125.$

TABLE 4.—(*Continued*)

Detailed Working Data	Load Point 3	Load Point 2	Load Point 1
W = material handled	11,000 lb.	11,000 lb.	5,500 lb.
W_s = skip	6,100 lb.	6,100 lb.	6,100 lb.
W_r = rope per side	4,855 lb.	3,913 lb.	1,008 lb.
r_1 = smallest working radius of reel	3.73 ft.	4.08 ft.	4.96 ft.
r_2 = largest working radius of reel	5.23 ft.	5.23 ft.	5.23 ft.
r_{ua} = radius up side end of acceleration	3.86 ft.	4.21 ft.	5.09 ft.
r_{da} = radius down side end of acceleration	5.10 ft.	5.10 ft.	5.10 ft.
r_{ub} = radius up side begin retardation	5.13 ft.	5.13 ft.	5.13 ft.
r_{db} = radius down side begin retardation	3.855 ft.	4.18 ft.	5.06 ft.
W_{ua} = wt. of rope wound on during acceleration	390 lb.	396 lb.	483 lb.
W_{da} = wt. of rope wound off during acceleration	497 lb.	497 lb.	497 lb.
W_{ub} = wt. of rope wound on begin retard	400 lb.	400 lb.	400 lb.
W_{db} = wt. of rope wound off begin retard	294 lb.	318 lb.	385 lb.
WR^2 — of motor armature	12,000 lb-ft.^{2}	12,000 lb-ft.^{2}	12,000 lb-ft.^{2}
WR^2 — of hoist and permanently wound rope	1,093,610 lb-ft.^{2}	1,122,530 lb-ft.^{2}	1,239,300 lb-ft.^{2}
Reel speed	0.875 r.p.s.	0.875 r.p.s.	0.875 r.p.s.
Efficiency (1 herringbone gear reduction	84.5 per cent	84.5 per cent	84.5 per cent
Time of acceleration	10 sec.	10 sec.	10 sec.
Time of retardation	8 sec.	8 sec.	8 sec.
Rope wrap increment (rope pitch)	0.03 ft.	0.03 ft.	0.03 ft.
Static Moments Load Point 3 (*Example*)	**Lb-ft.**	**Lb-ft.**	**Lb-ft.**
Up load U_1 (loaded skip) r_1 (11,000 + 6,100)3.73	63,783	69,768	57,536
Up rope R_{u1} rope × r_1 4,855 × 3.73	18,208	15,965	5,000
Down skip D_1 skip × r_2 6,100 × 5.23	31,903	31,903	31,903
Up load U_a (loaded skip) r_{ua} (17,100 × 3.86	66,006	71,991	59,044
Up rope R_{ua} (4,855 − 390)3.86	17,235	14,806	2,672
Down skip D_a (empty skip) r_{da} 6,100 × 5.1	31,110	31,110	31,110
Down rope R_{da} $W_{da} \times R_{da}$ 497 × 5.1	2,535	2,534	2,534
Up load U_b (loaded skip) r_{ub} 17,100 × 5.125	87,637	87,723	59,508
Up rope R_{ub} $W_{ub} \times r_{ub}$ 400 × 5.125	2,050	2,050	2,050
Down skip D_b (empty skip) r_{db} 6,100 × 3.855	23,515	25,498	30,866
Down rope R_{db} $(W_r - W_{db}) \times r_{db}$ (4,855 − 294)3.855	17,560	15,027	3,152
Up load U_2 (loaded skip) r_2 17,100 × 5.23	89,433	89,433	60,668
Down skip D_2 (empty skip) r_1 6,100 × 3.73	22,753	24,888	30,256
Down rope R_{d2} (rope) r_1 4,855 × 3.73	18,208	15,965	5,000

TABLE 4.—(*Continued*)

Accelerating Moments

Load point 3. (Example)	Load Point 3 Lb-ft.	Load Point 2 Lb-ft.	Load Point 1 Lb-ft.
Up load Up rope Head sheave			
$\frac{(11{,}000 + 6{,}100 + 4{,}855 + 1{,}100)2\pi(3.86)^2 \times 0.875}{32.2 \times 10}$	5,862	6,710	5,580
Down rope			
$\frac{\pi \times 4{,}855(3.73^2 + 5.23^2) \times 0.875}{32.2 \times 10}$	1,710	1,468	460
Down load Head sheave			
$\frac{(6{,}100 + 1{,}100) \times 2\pi \times 5.23^2 \times 0.875}{32.2 \times 10}$	3,360	3,360	3,360
Rotating parts			
$\frac{1{,}093{,}610 \times 2\pi \times 0.875}{32.2 \times 10}$	18,658	19,152	21,170
Total "A" moment..	29,590	30,690	30,570

Retarding Moments

Load point 3. (Example)	Load Point 3 Lb-ft.	Load Point 2 Lb-ft.	Load Point 1 Lb-ft.
Up load Head sheave			
$\frac{(11{,}000 + 6{,}100 + 1{,}100) \times 2\pi(5.23)^2 \times 0.875}{32.2 \times 8}$	10,640	10,640	7,420
Up rope			
$\frac{\pi \times 4{,}855 \times (3.73^2 + 5.23^2) \times 0.875}{32.2 \times 8}$	2,141	1,835	556
Down load Down rope Head sheave			
$\frac{(6{,}100 + 4{,}855 + 1{,}100) \times 2\pi \times 3.855^2 \times 0.875}{32.2 \times 8}$	3,824	4,185	4,484
Rotating parts			
$\frac{1{,}093{,}610 \times 2\pi \times 0.875}{32.2 \times 8}$	23,325	23,940	26,464
Total "B" moment..	39,930	40,600	38,924

Combined Moments

	Load Point 3 Lb-ft.	Load Point 2 Lb-ft.	Load Point 1 Lb-ft.
(*a*) Accelerating moment	29,590	30,690	30,570
(*b*) Retarding moment	−39,930	−40,600	−38,924
(*c*) $U_1 + R_{u1} - D_1 = 63{,}783 + 18{,}208 - 31{,}903$	50,088	53,830	30,629
(*d*) $U_a + R_{ua} - D_a - R_{da} = 66{,}006 + 17{,}285 - 31{,}110 - 2{,}535$	49,596	53,153	28,072
(*e*) $U_b + R_{ub} - D_b - R_{db} = 87{,}637 + 2{,}050 - 23{,}515 - 17{,}560$	48,612	49,250	27,542
(*f*) $U_2 - D_2 - R_{d2} = 89{,}433 - 22{,}753 - 18{,}208$	48,472	48,580	25,412
(*g*) Friction:			
$F = \frac{W(r_1 + r_2)}{2} \times \frac{1 - E}{E} = \frac{11{,}000 \times (3.73 + 5.23)}{2} \frac{(1 - 0.845)}{0.845} =$	9,042	9,380	5,150

Horsepower

	Hp. due to motor WR^2	Load Point 3	Load Point 2	Load Point 1
Hp. at "A" $\left(a + g + \frac{d + 2.c}{3}\right) \times \frac{2\pi \times 0.875}{550} = 885$	+100	985	1,036	755
Hp. at "B" $(d + g) \times \frac{2\pi \times 0.875}{550} = 586$		586	625	322
Hp. at "C" $(e + g) \times \frac{2\pi \times 0.875}{550} = 576$		576	586	306
Hp. at "D" $\left(b + g + \frac{e + 2 \times f}{3}\right) \times \frac{2\pi \times 0.875}{550} = 176$	−125	51	50	−202
Root-mean-square hp. required: At sea level				
$= \sqrt{\frac{(985)^2 10 + (586^2 + 576^2 + 586 \times 576)47.22 + 51 \times 8}{\frac{10}{2} + 47.22 + \frac{8}{2} + \frac{15}{4}}}$		655	704	662
For 8,000-ft. altitude, divide by 0.864		758	815	767

It was desirable to retain the 12-in. shaft, since the bearings, reels, clutch disks, etc., were all assembled with it, and there were spare hoists of this type on hand. The shaft's suitability was demonstrated as follows: Equating the torsional moment of the load to the moment of resistance,[10] $T = \frac{\pi d^3 S}{16}$; substituting the value of T from the horsepower equation in terms of r.p.m., hp. $= \frac{2\pi \text{ r.p.m. } T}{33000}$. Hence, $S = \frac{26700 \text{ hp.}}{d^3 \text{ r.p.m.}}$.

With peak power on the hoist (Fig. 11) we have 936 hp. for the end of acceleration from load point 2. Therefore:

$$S = \frac{26700 \times 936}{(12)^3 \times 52.5} = 476{,}000 \text{ lb. per sq. ft.} = 3300 \text{ lb. per sq. in.}$$

Since 30,000 lb. per sq. in. is a minimum S, the safety factor is 9.

Considering the stiffness of the shaft, the angular deflection, θ, should not be over 1° for a shaft length 20 times its diameter. That is, the allowable θ would be $\frac{183\frac{1}{2}}{20 \times 12} \times 1 = 0.765$ degree. The torsional modulus of elasticity E_t represents the ratio of the unit stress to the unit deformation, so

$$E_t = \frac{S \times \text{length in inches}}{\text{deflection in inches}} = \frac{SL}{X}$$

$$X = \frac{\theta \pi d}{360}, \text{ or } \theta = \frac{360 SL}{\pi d\, E_t}$$

since $S = \frac{16T}{\pi d^3}$ and $T = \frac{33{,}000 \text{ hp.}}{2\pi \text{ r.p.m.}}$, $\theta = \frac{584L\left(\frac{33{,}000 \text{ hp.}}{2\pi \text{ r.p.m.}}\right)}{d^4\, E_t}$

In this case $L = 183.5$ in. and E_t is about 30,000,000 lb. per sq. in., so $\theta = 0.194°$, which is well within the limit.[11]

The ultimate strength of the Roebling ⅝ by 5-in. plow-steel rope is 177,000 lb. The greatest strain on the rope is when the full load is being accelerated from the greatest depth, i. e., load point 3.

	Pounds	Pounds
Maximum ore load	11,000	
Skip weight	6,100	
Max. weight of rope	4,855	
Total static load		21,955
Bending stresses, 6 ft. sheave and 3.73 ft. reel		6,520
Max. acceleration stresses, at 0.0875 r.p.s. per sec. or 2.875 ft. per sec. per sec.		1,955

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[10] Where T = torque or torsional moment, S = unit stress, d = diameter.

[11] The corrections for keyways, as affecting shaft strength and stiffness, and according to the formula of Prof. H. F. Moore (*Bull.* 42, Univ. Illinois Experiment Station) do not materially affect the results as given here.

	POUNDS
Friction over sheave	610
Total	31,040

therefore $\frac{177{,}000}{31{,}040} = 5.65$ safety factor in rope.

The brakes and clutches were lined with special asbestos blocks containing brass wire reinforcement. Their coefficient of friction is 0.40. The greatest moment to be held by the brakes is the static load when the loaded skip just reaches the dump: $17{,}100 \times 5.23 = 89{,}435$ ft-lb. Considering this as unbalanced for the most severe condition, with the radius of the brake as 5.25 ft., the tangential load is: $\frac{89{,}435}{5.25} = 17{,}031$ lb. Dividing by the coefficient of friction, $\frac{17{,}031}{0.40} = 42{,}577$ lb. to be applied to brakes. Each brake has a projected area of 920 sq. in., so if this load is to be sustained with two brakes, assuming the clutch on unloaded side released, we have 21,289 lb. per brake beam or 23 lb. per sq. in.; or, for balanced load and allowing 10 per cent for friction, 17.2 lb. per sq. in. The brake beams had been operated by air cylinders. These were replaced with the new Wellman-Seaver-Morgan quick-acting cylinders. The connections from the operator's hand lever are such that the piston stops and automatically locks itself in any position corresponding to that of the hand lever. The brake cylinders have poppet valves, 6-in. bore, 16-in. stroke, and operate on 80 to 100-lb. air pressure.

Because of the hoist's heavy duty it was provided with single-reduction herringbone gear, part of which installation was made in the company's shops. The pinion shaft was coupled to the motor by a Francke heavy-pattern flexible coupling.

The Welch safety device on the old hoist was replaced by Lilly model D controllers. Each reel is geared to a controller, which automatically cuts off power from the motor and applies the brakes in case of overspeed during acceleration, full load travel, or retardation, or in case of overwind, or failure of line current. Track limit switches, placed on the guides slightly above the dump and connected with the controllers, prevent overwind in case the setting of the Lilly cams is incorrect. There are also cams for acceleration and retardation, shaped to effect control in accordance with the cycle determined. If the engineer should start too quickly or not slow down fast enough, the Lilly controller rings a warning bell; if the speed is not corrected at once, power is thrown off and the brakes applied. Another bell attachment rings just before the period of retardation is reached. These mechanical safety devices are of great importance at the San Juan Pachuca shaft, because the clearance between the rope clamps and the sheaves is only 11 ft. when the skip is in dumping position.

ELECTRICAL ENGINEERING

A wound-rotor type of induction motor was selected for the hoist, as alternating current only was available for large demands, and the greater cost of a flywheel set was not warranted. Full reversible magnetic control with grid resistances was chosen because it requires less attention and maintenance than liquid rheostatic control. The latter is best suited to partial speed operations, which would rarely be required for hoisting ore. Both the primary and secondary contactors of the plant are controlled through a master controller, which furnishes only sufficient current to operate the magnetic solenoids.

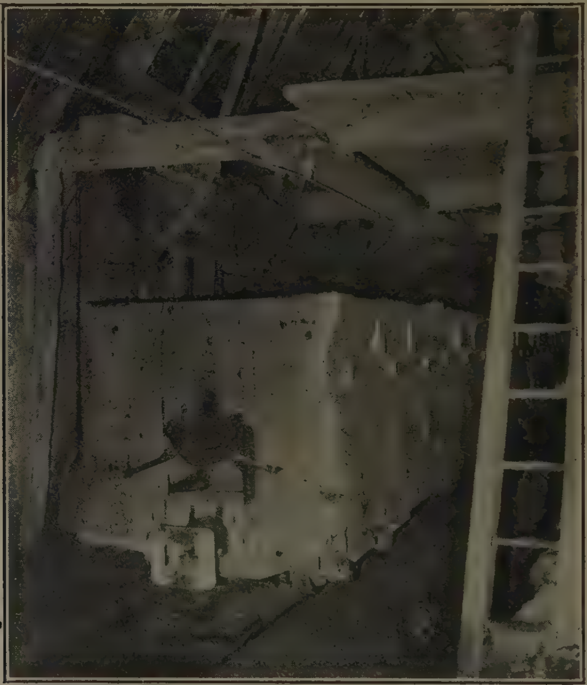

FIG. 12.—LOADING TRAIN ON MAIN HAULAGE FROM GASTINEAU GATES AT LA RICA.

The direction of rotation in the motor is controlled by reversing the line connections in its primary circuit. This is done with triple-pole, double-throw contactors, functioning on the three motor leads and provided with mechanical and electrical interlock. The contacts break the circuit in air and have magnetic blowouts and arc chutes to dampen the arc and insure its positive rupture. Protection is provided against closing one set of contactors until the arcs are completely ruptured at the tips of the others.

The secondary control consists of three-phase resistors, connected across the collector rings of the motor in its rotor circuit. Secondary contactors, closing in successive steps, short-circuit the resistors in the rotor circuit. The contactors are controlled from the master controller on the operator's platform. Automatic acceleration is provided by current limit relays, which cut out successive sections of resistance when the current falls to a predetermined minimum. This is done in eight steps. As each step requires the current to fall below the predetermined point, the hoist comes up to full speed automatically when the master controller is thrown to full-speed position. The operator can start as slowly as desired, as the first six steps are controlled individually by notches on the master controller before the full speed position is reached. There is sufficient ohmic capacity in the rheostat to limit the motor torque to a low value on starting and on reversal. This permits slow speeds with light loads when required.

The master controller is of the drum type, with a vertical operating handle. A push button closes the accelerating contactors, so that full starting torque can be obtained for emergencies. A back-out switch is placed within reach of the operator, by which he can jog the skip back if it passes the track limit switches. Two under-voltage contactors are mounted on the contactor panel so that, in case of power failure, the hoist cannot be started until the master controller is in the "off" position.

The preliminary calculations showed that a 750-hp. hoist motor was required. The General Electric Co. quoted on 750 and 850-hp. motors, 40° C. rise at 7500 ft. The 850-hp. motor was finally chosen, as it gave a 13 per cent margin of safety over the calculations at 5 per cent increase in cost. As 50-cycle current was available, a 16-pole motor was chosen for a full-load speed of 365 r.p.m. This gave the most suitable gear reduction, considering the space for installing the gear and pinion.

The specifications of the motor are:

Voltage	1,040
Synchronous speed, r.p.m	375
Full-load speed, r.p.m	368
Phases	3
Cycles	50
Horsepower	850
Temp. rise at full load continuously, at 7500 ft. altitude	40° C.
Approx. net weight, lb	15,300
Approx. rotor current, full load, amp	495
Max. running torque, lb-ft	27,500
Efficiency 4/4 load, per cent	93.5
3/4 load, per cent	93
1/2 load, per cent	92
Power factor 4/4 load	86
3/4 load	83
1/2 load	76
Height, in	73
Width over bearings, in	66
Length, in	73

Installation and Operation

Eight days were required to dismantle the old hoist, assemble the new equipment, set the new motor and line up all shafts and gears. The reel hubs were wound with old ½ × 5-in. flat rope, to bring them up to the minimum diameter indicated by the calculations. The miscellaneous work of setting the control apparatus and motor connections, and making alterations to the headframe and Kimberley dump, was done while the hoist was being remodeled.

The hoist has been in operation four years. The only serious mechanical trouble was heating of the pinion shaft bearings, probably due to the use of a 9-in. shaft. As this could not be easily changed, a center bearing was put in and all three bearings fed by a small oil pump; this has proved satisfactory. To check the operation of each hoist engineer a Bristol recording revolution-counter was installed. A magneto attached to the end of the hoist shaft operated this device, which contained a slow-geared telechron clock mechanism, feeding the paper at ½ in. per min. This was used for only a few months, as the hoist engineers soon became adept in their work with the assurance of safety from the Lilly controllers.

The hoist keeps the mill supplied with 123,000 to 130,000 dry short tons per month, without working Sundays; 6300 tons have sometimes been delivered during a 24-hr. run. The record was 6335 tons in 24 hr., of which 18 hr. 45 min. was hoisting time, the remaining 5 hr. 15 min. being used for changing ores and load points, cleaning skips, etc. Of this tonnage, 17 per cent came from load point 1, 69 per cent from point 2, and 14 per cent from point 3. The actual hoisting time is more than three hours less than the theoretical time required for this work, because the loading time is usually 10 sec., instead of 15 sec. as calculated (sometimes as low as 8 sec.), and the motor speed at full load is 368 r.p.m., instead of 365 as calculated. Moreover, the skips from load point 1 average 3.45 tons instead of 2.75 tons as assumed. As this heavier duty requires more power than originally calculated, it is fortunate that the 850-hp. motor was purchased.

Conclusion

The centralization project was completed according to schedule and has given good working service from the start. The cost was heavy, even though all the expected benefits have been realized. The drastic decline in the price of silver since the undertaking was begun could not have been foreseen, but has made the centralization of much greater value than any calculation of cost reduction would have indicated. Due to the decline in the silver market, the workable ore reserves in the Pachuca district have been greatly reduced and nearly all new discoveries of importance have been in Real del Monte. In 1929, La Rica tramway was working to capacity. Had it not been for the centralization, giving flexibility of ore delivery, the Loreto mill would not have had sufficient ore for its demands, whereas the Guerrero mill would have been greatly oversupplied. The company is fortunate in having now a system of ore delivery which will allow the production schedule for each mine to follow the requirements of development and the desires of the management.

Particular credit is due to the management of the Compañía de Real del Monte y Pachuca for their far-sighted policy in sponsoring the centralization project. Mr. M. H. Kuryla, Managing Director, and Mr. E. L. Young, Director, were concerned at all times with the numerous problems arising after the general plan of work had been approved. Mr. John T. Lewis, Jr., Foreman in charge of mine construction, kept all parts of the work moving in their proper relation. His tireless attention to detail was largely responsible for carrying on the work according to schedule.

DISCUSSION

(*Lucien Eaton presiding*)

L. Eaton,* New York, N. Y.—Two or three things in this paper struck me as rather unusual in recent practice. One was the fact that 500 volts was used on underground haulage, which is rather uncommon in the United States. It is, however, quite common in Europe and is used almost exclusively on the Rand in South Africa, where they seem to have no difficulty from electrocution. It has a great advantage, of course, in a long haul of this kind.

Another thing was the use of a reel with flat ropes. I thought that reels and flat ropes were part of ancient history, like the Cornish pump. Probably they would not have used it without good reason, because anyone who has used the flat rope knows the difficulties of keeping it together. I know of one installation where all of one man's time was spent in lacing the strands.

There are some very interesting calculations as to horsepower required, voltage drop and so on, and hoist capacity. These are valuable in checking the manufacturers. They are, however, always inconclusive, because, as in this case, they are based on an assumption that is proved by experience to be wrong. For instance, the author makes elaborate calculations of horsepower required on locomotives, basing his assumption on rolling friction of 30 lb. per ton. Also, of course, he does not know the size of the copper that is in the windings of the motors, which will have a great effect on the adequacy of the cooling But throwing that out, he says that afterwards they found their rolling friction was 15 lb. per ton; in other words, 50 per cent of their assumption. That being the case, why go to the elaboration of calculation when your assumption in the first place is a guess? It reminds me of Professor Johnson's famous analysis of the situation made some 30 years ago on the strength of retaining walls. There are three formulas that are used to calculate the strength of a retaining wall, but in each case you must assume one element, either the point of application of the force, its direction or its amount. He said, "As long as you have to make one guess, why not guess the thickness of the retaining wall in the first place?"

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* Consulting Mining Engineer.

C. F. Jackson,* Washington, D. C.—It would be interesting to know what the approximate saving effected by this concentration amounted to. Possibly the changed conditions after the installation went into effect would make it difficult to tell exactly what the saving was, but I think an approximate idea would be useful.

I was impressed with the fact some years ago, in making up figures on costs at some 60 or 70 mines, that the average transportation cost, including the hoisting at those mines, was 20 per cent of the total underground cost. In one mine in the list, the transportation cost amounted to 56 per cent of the total mining cost. I imagine that with the overland transportation involved here, the saving might have been impressive.

F. S. Mulock,† Boston, Mass.—The new haulage system together with the centralization of milling helped materially in reducing costs. Other factors contributed also.

J. H. Steinmesch,‡ Rolla, Mo.—Was the very large storing between levels considered an important part of the transportation system?

F. S. Mulock.—The large storage capacity underground was necessary to supplement the very limited storage capacity available at the mill, the latter capacity being limited by lack of space. In practice, the hoisting, crushing and sampling operations are coördinated in such manner as to lengthen as much as possible the period of hoisting each of the various ores. Had it not been necessary to keep the ores from the various sources separate until sampled, the provisions for underground storage could have been greatly simplified.

H. I. Altshuler (written discussion).—The 500 volts was selected for the main level haulage because of its obvious advantages, the danger of electrocution being small in this large section where the only work to be done was ore transport. We have had no trouble so far.

The principal reason for adopting flat ropes was to utilize the hoist that was actually in service when the centralization took place. We have to employ a rope expert and small crew to keep up the service on the many hoists in operation. The continued use of flat rope at San Juan Pachuca did not require any increase in this crew, even though that service is severe. The flat ropes on the San Juan Pachuca ore hoist require resewing every 3½ to 4 months and their life before discarding is 2½ to 3 years. We sew our ropes with 42 in. between double stitches, or ties, whereas most factory ropes are furnished with 10 ft. 4 in. between ties.

Concerning Mr. Eaton's comment on the calculations, it might be pointed out that the calculations fall under two headings; i.e., those for the hoist and those for the locomotive.

The hoist calculations are standard practice and really require no comment. As a matter of fact, our practice lived up to these calculations very closely until we found that our equipment was sufficiently strong to stand a slightly heavier duty. This duty was then applied and a somewhat greater capacity has been attained than those indicated by the calculations, as explained in the paper. The present solution of the hoist problem is more satisfactory than the one proposed by the manufacturers, and had not these calculations been made on the job we would have been committed to an installation less suitable to our conditions.

Concerning the locomotive heating calculations it is true that the method used is not theoretically accurate; the exact calculation, taking into consideration all electrical conditions, is very involved and laborious. The calculations as given in this paper are relatively simple and can be made in a short time. The size of the copper in the windings is immaterial in this case, since the heating curve provided by the manufacturers takes into account the physical characteristics of the motors being considered, and this curve is used for these calculations. It is true that 30 lb. friction for the trail load was allowed in this case and that a much lighter rolling friction was actually realized. The reason that the calculations were made and that 30 lb. rolling friction was taken was the fact that the rolling friction of the cars was not known at the time the calculations were made, and it was decided to have a large margin of safety for this haulage. It was realized that 30 lb. was a high figure, which could not possibly be exceeded, and for this reason the calculations were made to give assurance that no overheating would take place under the worst possible condition. This method of calculation, as described more fully in the footnote reference in the paper, agrees fairly closely with actual practice when the rolling friction of the load and characteristics of the locomotive are known. In this case the calculations served a useful purpose in allowing us to purchase equipment that we knew to have ample capacity for the worst possible condition that could exist on this haulage.

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* U. S. Bureau of Mines.

† Engineer, U. S. Smelting, Refining and Mining Co.

‡ Professor of Mining, School of Mines and Metallurgy, University of Missouri.

The Owyhee Tunnels

By Pierre R. Hines, Portland, Oregon

(New York Meeting, February, 1933)

Abstract*

A description of the methods used in driving two tunnels, 2 miles long each, one a 9-ft. and the other a 19-ft. diameter section, for the Owyhee Irrigation Project, Oregon. One heading of the smaller tunnel was driven 1315 ft. in 73 shifts in one month and 10,966 ft. of this section completed in 11½ months using two shifts for eight months and three shifts the rest of the time. This is a good illustration of the proper use of a mechanical shovel and the importance of careful selection of supplementary equipment to secure best results. Large cars, speedier and heavier locomotives, 36-in. gage track of 45-lb. rail, and a good switching method increased actual shoveling time and cut down muck removal period about 45 minutes from average. The larger tunnel, after trying out standard electric revolving shovels of both the rail mounting and caterpillar tread types, was excavated with a mechanical mucker. This is the first time that a tunnel of so large a section has been handled this way. This record work was accomplished without drill carriages.

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* The paper was published in *The Explosives Engineer*, December, 1932 and January, 1933, as A.I.M.E. *Contribution* No. 1.

Sinking and Equipment of No. 2 Shaft at Minas de Matahambre

BY DUDLEY D. HOMER* AND ROBERT H. CROMWELL,† MATAHAMBRE, CUBA

(New York Meeting, February, 1933)

ABSTRACT‡

THE Minas de Matahambre No. 2 shaft, vertical, four-compartment lined with steel sets and hardwood lagging, has been sunk 2057 ft. On account of rock structure, over-all cross-section dimensions are 23 by 6 ft. Certain sections have been concreted. Two compartments are for handling ore and men, one for supplies and men, and one for manway, cables and counterweight. Two 250-ton capacity skip pockets with air-operated gates and steel cartridge pockets have been installed underground. A steel "A" headframe, a 250-ton steel receiving bin, and skip-to-man-cage changing equipment have been provided. Skip capacity is 102½ cu. ft. The hoist is a Nordberg double-drum 10 ft. by 6 ft. 6 in. geared, electrically driven, with parallel grooving for 1⅜-in. cables. Maximum hoisting speed is 1600 ft. per minute. The motor-generator set, control panel, slip regulator and hoist accumulator are housed in the basement of the hoist house. The electrical equipment is General Electric. The direct-current hoist motor is 800 hp. Control is of the Ward-Leonard-Ilgner type. The motor-generator flywheel set, normal speed 720 r.p.m., consists of an 800-hp. induction motor, a 22-ton flywheel (dia. 117 in.) and a 650-kw. generator.

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* General Manager, Minas de Matahambre S. A.

† Assistant General Manager, Minas de Matahambre S. A.

‡ The paper was published in *The Explosives Engineer*, February, 1933, as A.I.M.E. *Contribution No.* 2.

Ground Movement from Mining in Brier Hill Mine, Norway, Michigan*

By George S. Rice,† Washington, D. C.

(New York Meeting, February, 1934)

A problem of possible subsidence of the surface from mining operations, which might have had disastrous results, arose in 1913 at the Brier Hill mine, of the Penn Iron Mining Co., near Norway, Mich., now managed by Pickands, Mather & Co. for the Bethlehem Steel Co. A double track and sidings of the Chicago & North Western Railroad passed over part of the mine, and if subsidence had reached the surface it might have affected the stability of the tracks.

The chief purpose of this paper is to record the facts disclosed by investigations of underground movement from 1913 to 1919, when the tracks were moved, and subsequently at intervals, as engineering data of value from mining carried on below the tracks in the steeply pitching lenses.

A secondary purpose of this paper is to describe a novel method for determining the rate of speed of upward breaking above the mine excavations. This method was found effective and might be used elsewhere under similar conditions.

The mine workings under consideration were developed from the Brier Hill shaft, which was circular, concrete-lined, 14 ft. in diameter and about 900 ft. deep. They extended about 800 ft. westward from the shaft, and nearly parallel to a corresponding length of the railroad tracks. The highest part of these workings was at a depth of 400 ft. below the tracks (Figs. 1 to 12). The mining company had the right to remove the ore, the railroad assuming that the surface would be supported. Finally, without legal controversy, an amicable adjustment was reached as to what should be done to lessen or obviate the hazard of subsidence.

Method of Mining

Before the problem became acute, the mining company, to decrease the possibility of subsidence, had been mining the part of the orebody that lay under the tracks by a room-and-pillar system, back-filling with

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Manuscript received at the office of the Institute Nov. 29, 1933.

* Published by permission of the Director of the United States Bureau of Mines

† Chief Mining Engineer, United States Bureau of Mines.

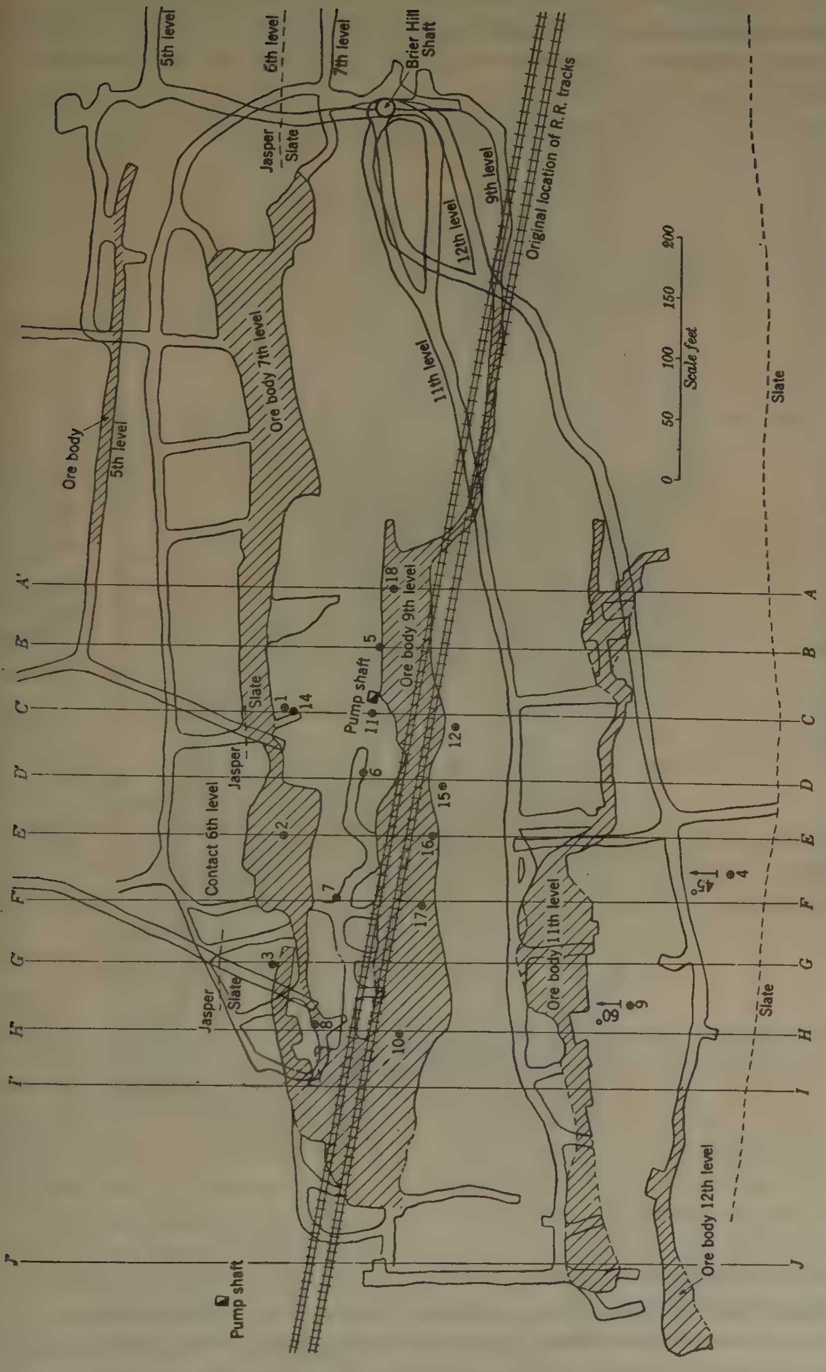

FIG. 1.—PLAN VIEW, BRIER HILL MINE.

Shows main orebody on seventh, ninth, eleventh and twelfth levels, main-line railroad tracks, top of 17 diamond-drill holes and lines of cross-sections *A-A'* to *I-I'*.

waste rock and gravel. The rooms were opened by first taking out a horizontal slice from foot to hanging wall, followed by 7-ft. slices above. The usual square-set timbering was employed; the ore blasted down filling the excavations, except for the chutes, ladderways and airways. Excess ore was dropped into a loading chute at the haulage level.

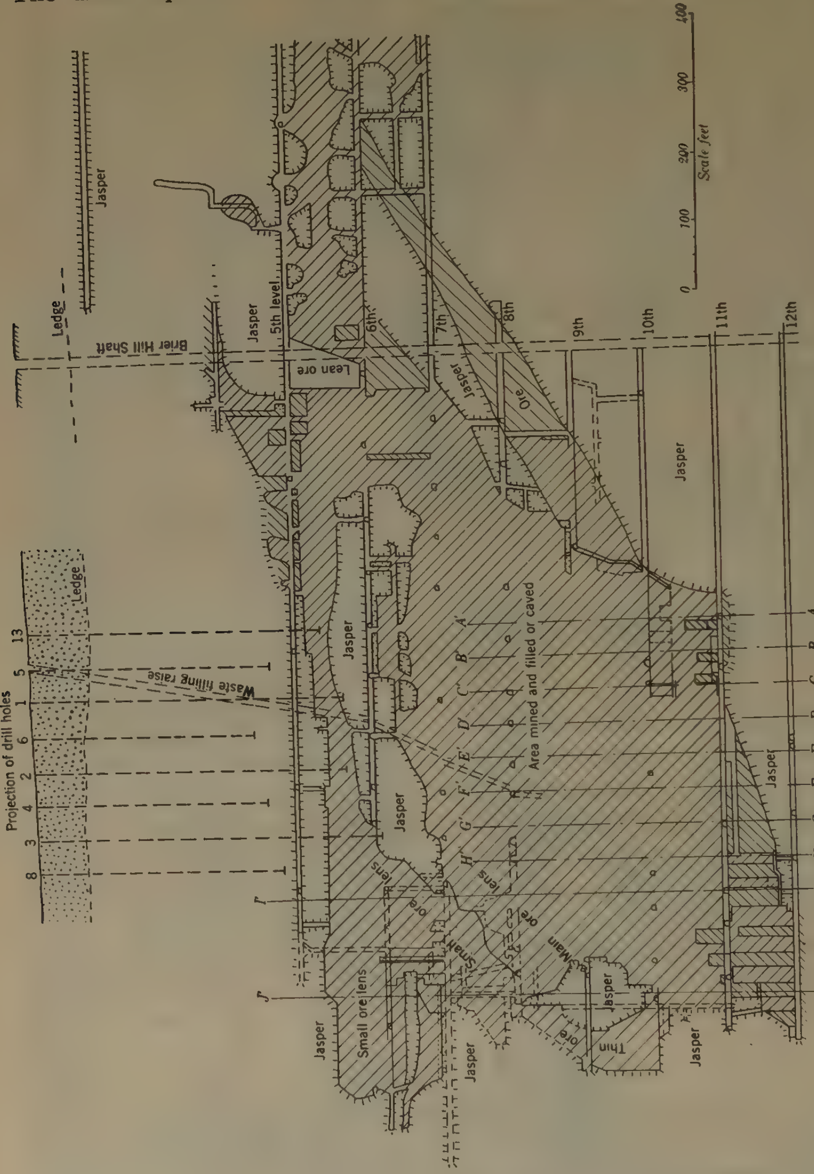

FIG. 2.—LONGITUDINAL SECTION OF BRIER HILL MINE, SHOWING AREA MINED OUT OF PRINCIPAL OREBODY AND ORE REMAINING DECEMBER 31, 1921.

Also levels, crosscuts, winzes and position of eight of the diamond-drill holes; also lines of cross-sections *A* to *J*. The main ore lens is superimposed on the small ore lens in the projection at the upper left-hand corner.

After three or four slices had been taken from a room, the ore was drawn off, and waste-rock filling run into the square sets from an upper level or from the surface, through an inclined winze. Then other slices were similarly taken out above, the ore drawn off and filling done up to within 10 or 15 ft. of the level above. When several rooms had been finished, the 22-ft. pillars between them were similarly attacked, beginning at the bottom; back-stoping horizontal slices from foot to hanging wall, with square-set timbering. On reaching the top of the pillar, the sets were filled with waste and gravel, as had been done in the rooms.

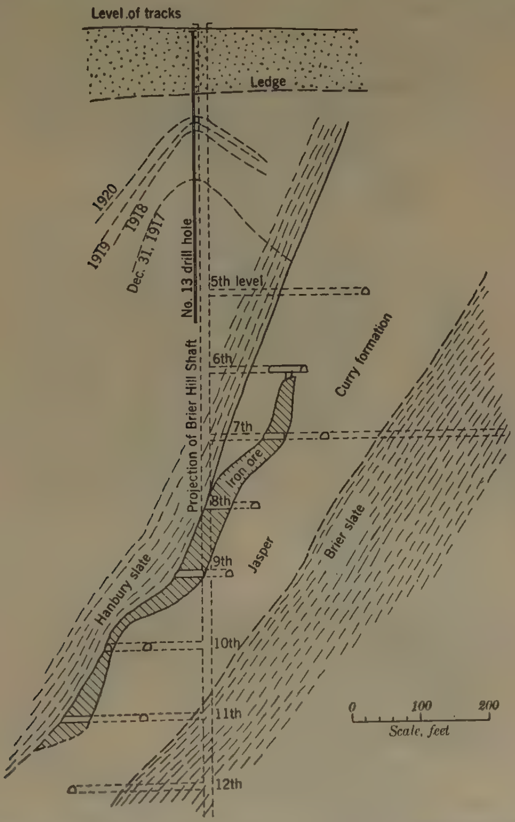

Fig. 3.—Brier Hill mine, cross-section *A-A′*, 400 ft. west of shaft. Showing main orebody in Curry formation, railroad level and diamond-drill hole No. 13, which disclosed successive caves or breaks. Shape of breaks on either side of hole, indicated by Janson, is conjectural.

By November, 1913, the lenslike orebodies below the fourth level had been worked as stated above, although from the seventh level to the ninth all the pillars had not been removed. The pillars in the eighth and ninth levels in the principal orebody having shown the effects of pressure accompanied by squeeze of timbers, the management deemed it hazardous to the miners to extract the pillars from the bottom up and decided to mine them by top slicing downward, with close timbering; then shoot out the timbers on completing a slice, thus forming a timber mat. By this system, extensively used in the Michigan iron mines, the waste resting on the timbers follows down on the mat. Although the resulting caving would usually keep within the walls of so steeply dipping a vein, provided the walls are strong as they appeared to be in this case, the management

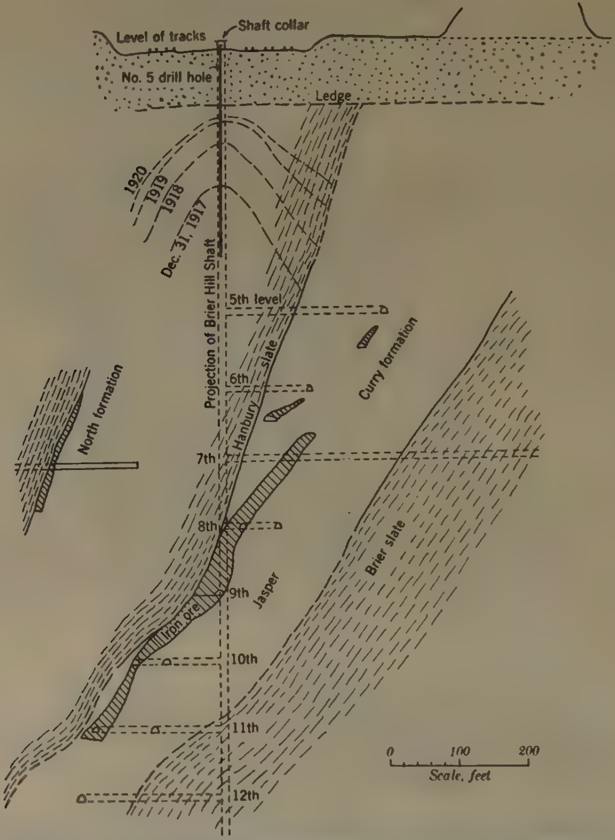

Fig. 4.—Brier Hill mine, cross-section *B-B′*, 450 ft. west of shaft. Showing orebodies, railroad level, diamond-drill hole No. 5, which disclosed successive caves or breaks. Shape of breaks conjectural.

feared that caving through the hanging wall might also occur and on reaching the drift formation cause sudden dropping of the railroad tracks. The railroad company, alarmed at this possibility, arranged with the mining company, represented by William Kelly, then general manager, to appoint a joint commission to make recommendations. The late Prof. F. W. Sperr and the writer were asked to serve; the latter enlisting the assistance of the late Edwin Higgins, of the Bureau of Mines in the Lake Superior district.

Geological and Mining Conditions

The ore-bearing formation, known as the Curry or South, in which the two lenses under consideration occur, is 150 ft. thick at the Brier Hill mine, with jasper gangue rock. Its strike is N.75W., and the dip from 60° to 70° to the south (Figs. 3 to 12). The footwall is slate, locally termed Brier slate; the hanging wall of Hanbury slate, the lower layer of which is more graphitic and softer, tending to break small, and having a gouge or selvage 6 to 9 in. thick at the contact with the ore.

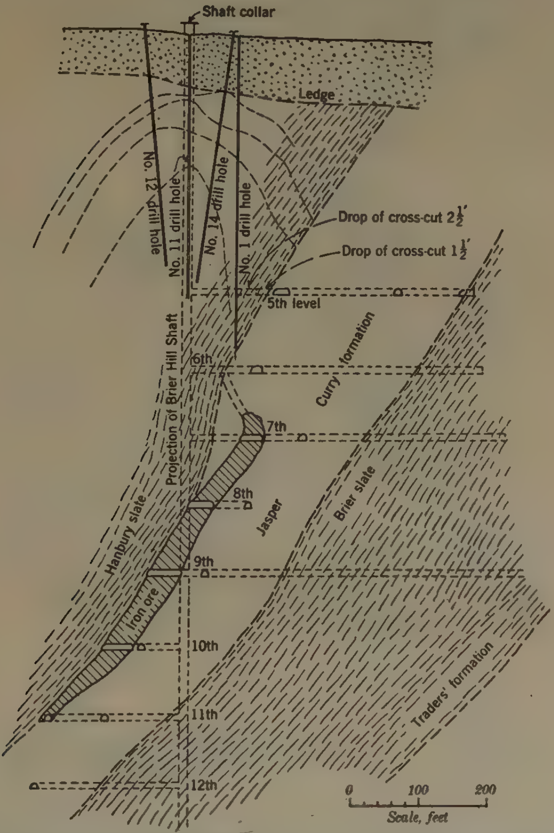

FIG. 5.—BRIER HILL MINE, CROSS-SECTION *C-C'*, 500 FT. WEST OF SHAFT. Showing main orebody, railroad level and diamond-drill holes 12, 11, 14 and 1, and crosscut on fifth level, which disclosed caves or breaks. Shape of breaks on either side of drill holes is conjectural.

The orebodies under discussion are in two lenses, the higher and smaller one near the footwall between the fourth and seventh levels; the large lens, nearer the hanging wall, extending from the sixth level to a then unknown depth below the tenth. The smaller lens, which had jasper both for the immediate hanging and footwalls, was very irregular, averaging about 10 ft. in width, but attaining 30 ft. at one place. The larger lens had a jasper hanging wall for the most part, but in places touched the slate hanging wall of the formation; its footwall was jasper throughout. This lens varied much in dip, cross-section, and thickness; its thickness averaged 30 to 40 ft., with a maximum of about 80 ft., and was about 800 ft. long.

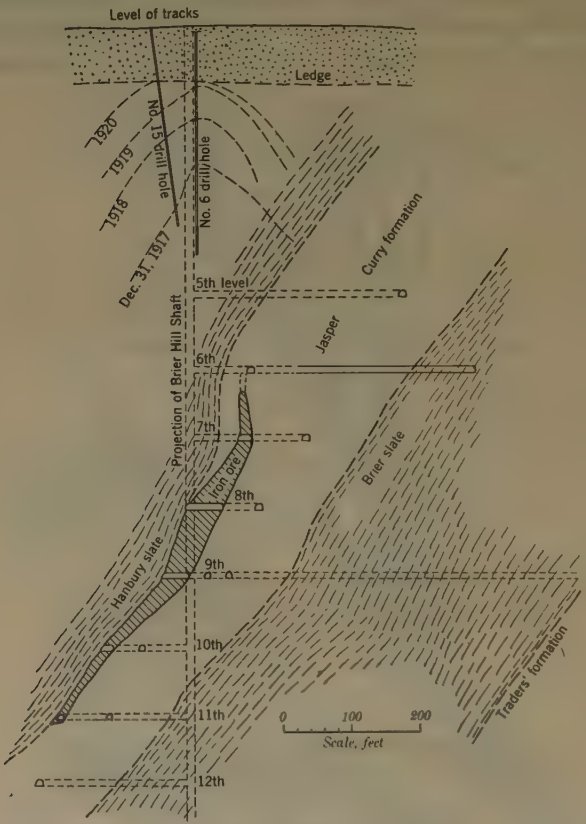

FIG. 6.—BRIER HILL MINE, CROSS-SECTION *D-D'*, 550 FT. WEST OF SHAFT. Showing main orebody, railroad level and diamond-drill holes 15 and 6, which disclosed successive caves or breaks. Shape of breaks on either side of drill holes is conjectural.

The mining of the smaller lens, because of its position near the footwall and not directly below the tracks, was not expected to cause subsidence. It was the mining of the large lens that raised that question, since it was not known then how deep below the tenth level the orebody extended. It was surmised, however, from the showing in the tenth level (900 ft. deep) that it would maintain its lateral dimensions to much greater depth; in which case, by mining it by top slicing, it was thought that surface subsidence might occur.

The tops of the steeply dipping ore formations were planed off by glacial action more or less horizontally at 60 to 75 ft. below the present undulating surface; under the railroad tracks the outcrop was covered with about 75 ft. of clay, sands and gravel. While excess water had been largely drained by operations in an adjoining mine, it was thought that these unconsolidated materials might still hold enough water to cause the sands to run into any wide break or cavein of the rock strata below, as had happened for example at the Chapin iron mine, where the railroad tracks required a high fill and continued grading over the subsiding ground.

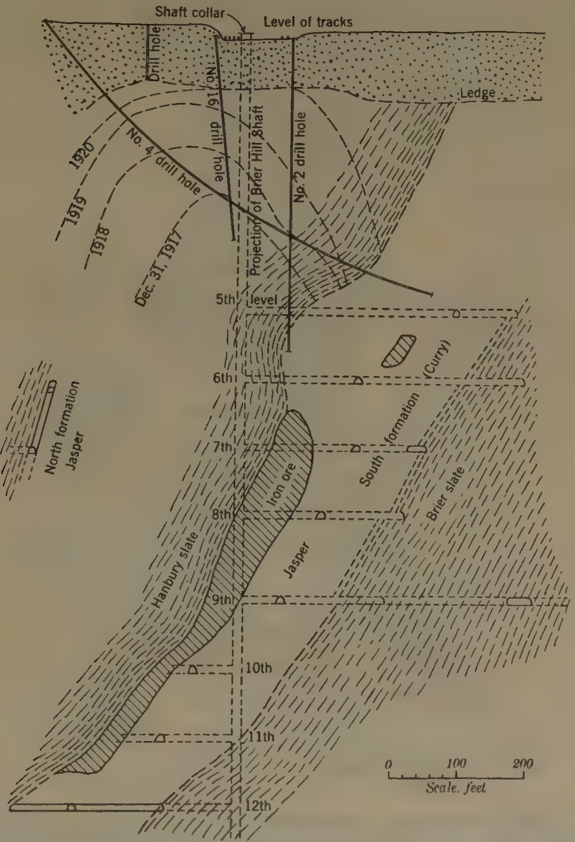

FIG. 7.—BRIER HILL, CROSS-SECTION *E-E'*, 600 FT. WEST OF SHAFT. Showing orebodies, railroad level and diamond-drill holes 4, drilled diagonally, 16 and 2 which disclosed successive caves or breaks, also one hole through drift to ledge only. Shape of breaks on either side of drill holes is conjectural.

Relation of Location of Railroad Tracks to Mine Workings.—The typical sections shown in Figs. 3 to 12 show that the tracks were nearly vertically above the middle of the large lens, but the contact planes of the slate hanging wall intersected the surface 200 to 300 ft. north of the tracks. This raised the question already mentioned: if breaks in the Hanbury-slate hanging wall occurred, would they be vertical or tend to follow up the steeply inclined ore formation to its outcrop under the drift? That is, whether the Hanbury slate would break diagonally across its bedding planes or remain intact, partly supported by the adjacent jasper pillars, the waste filling previously run in, and the spalled jasper and slate from the hanging wall.

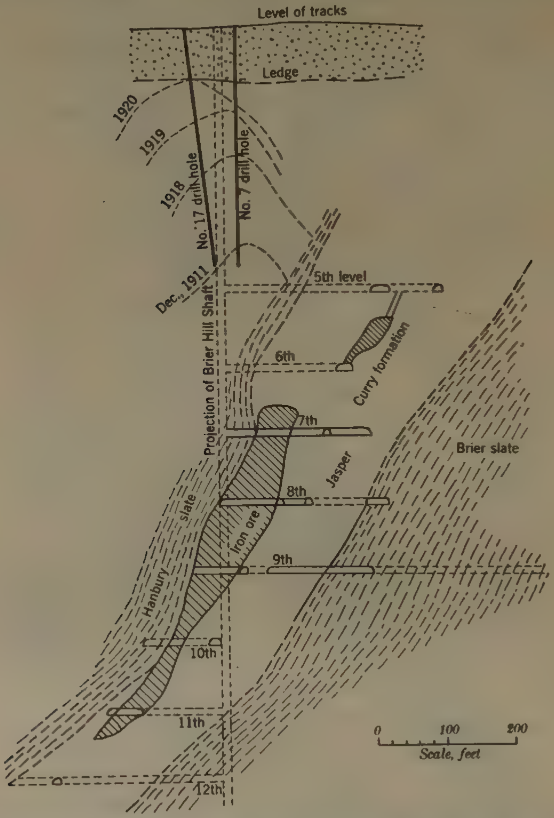

FIG. 8.—BRIER HILL MINE, CROSS-SECTION *F-F′*, 650 FT. WEST OF SHAFT. Showing orebodies, railroad level and diamond-drill holes 17 and 7, which disclosed successive caves or breaks. Shape of breaks on either side of drill holes is conjectural.

CONCLUSIONS OF REPORT OF COMMISSION

The report (dated Dec. 1, 1913) after summarizing the Commissioners' observations stated in substance that caving resulting from deep excavations of limited thickness may not reach the surface; because broken rock occupies 1¼ to 1½ greater volume than the solid rock and natural arches tend to form over the excavations. These arches buttress against the walls of the space above the caved material. Broken rock from the underside of the arches rolls against the side walls, and gradually lessens the unsupported span.

However, it was held tentatively that, even if extensive caving did not reach the top of the solid rock, cracks might open of sufficient size to permit water-bearing sands to run in and fill the voids of the fallen material. If so, sink holes in the surface would probably form. As to whether the caving would be confined within the walls of the ore formation, and so reach the outcrop considerably north of the railroad tracks, the Commissioners suggested, in view of a surface pit observed east of the West Vulcan shaft, that the hanging side of a cave in the Hanbury slate would probably be vertical.

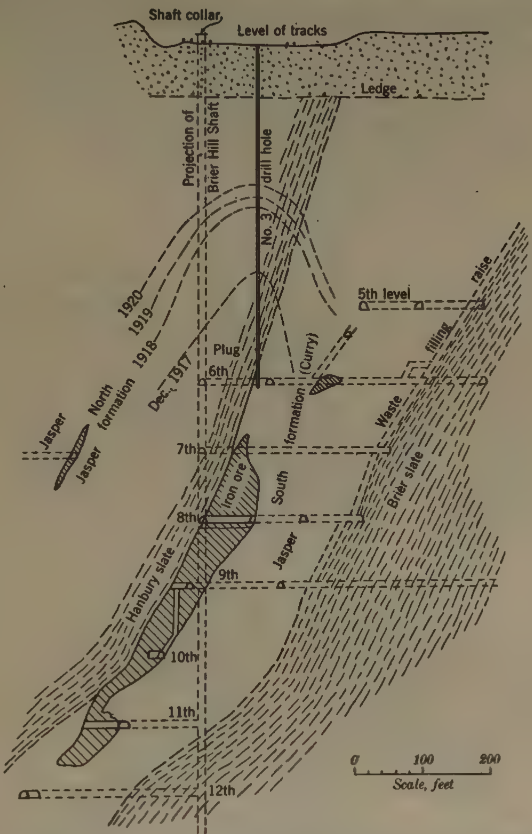

Fig. 9.—Brier Hill mine, cross-section *G-G′*, 700 ft. west of shaft. Showing orebodies, railroad level, waste filling raise and diamond-drill hole 3, which disclosed successive caves or breaks. Shape of breaks on either side of drill hole is conjectural.

The Commissioners' conclusions were: (1) "the removal of the pillars by top-slicing could probably continue with safety, provided the tracks are removed within a reasonable time—say one year[1] from date; (2) careful inspection of the surface should be maintained until the tracks are removed."

Discussion of Problems Subsequent to Report of Commission

In a conference between the chief engineer of the railway and the writer of this paper, subsequent to submission of the Commission's report, the former objected to moving the tracks, if this could be avoided with safety, as it involved a large outlay. The topography required moving to a considerable distance and new grading and construction would be required for several miles of double track.

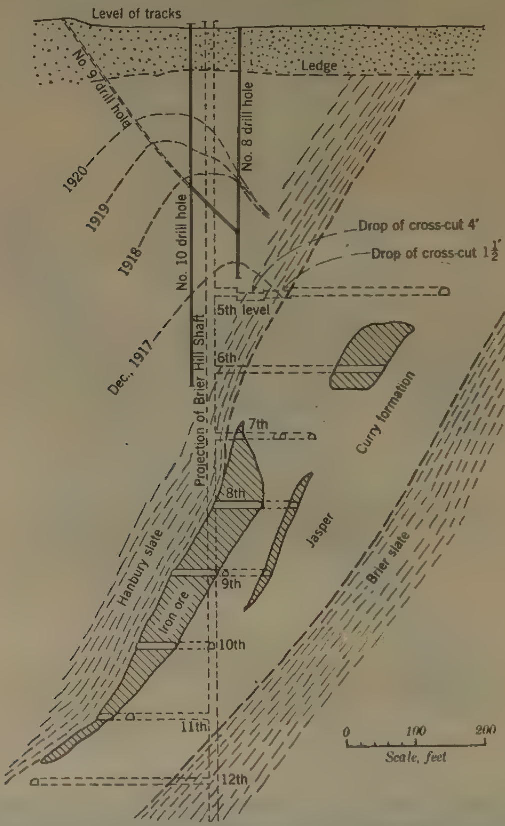

Fig. 10.—Brier Hill mine, cross-section *H-H'*, 750 ft. west of shaft. Showing orebodies, railroad level, drill holes 9 (diagonal), 10 and 8; also crosscut, which disclosed successive caves or breaks. Shape of breaks on either side of drill holes is conjectural.

The writer then submitted a memorandum giving alternative proposals of procedure for consideration.

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[1] The writer registered a minority objection to this limiting time.

1. To determine by drilling holes from the surface, as discussed in detail later, the rate of the caving as it extended upwards and in this way to obtain information in time to shift the tracks before the hazard became menacing.

2. While this testing was going on that one of several alternative filling methods be considered:

a. Driving crosscuts through solid jasper gangue from a footwall level at an elevation above the conjectured top of the cave, then after drilling holes in fan shape from the respective crosscut in order to find the position of the cave, to blast down inclined holes or chutes for running in waste rock, and (or)

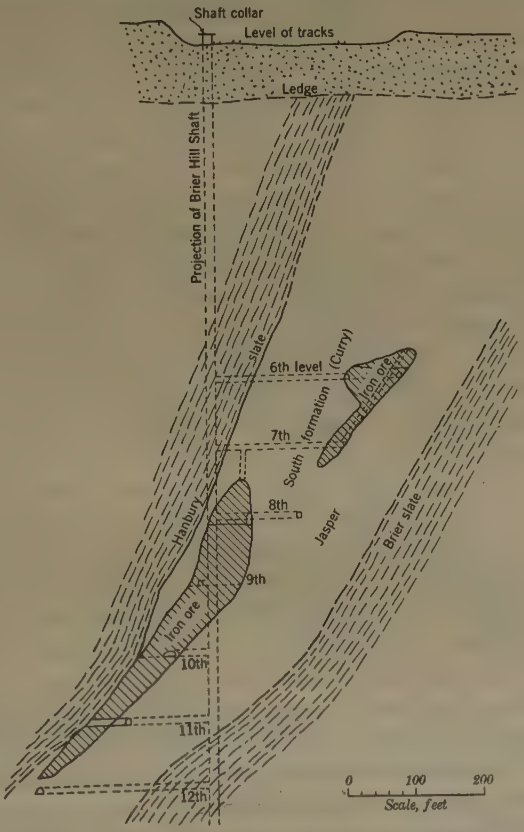

FIG. 11.—BRIER HILL MINE, CROSS-SECTION *I-I'*, 800 FT. WEST OF SHAFT.

b. Bore holes from the surface to the cavity or cavities and run in by water, sand and gravel following the hydraulic stowing system so extensively employed in German mines. This plan had been previously proposed but Mr. Kelly feared that the water used might injure the ore below.

3. Test the unit volume to which broken rock from a formation will compress under unit loads equivalent to those resulting from different depths of overburden in a large compression testing machine[2] to give data on which to base calculations whether the caving ground would self-fill before the caving reached the ledge.

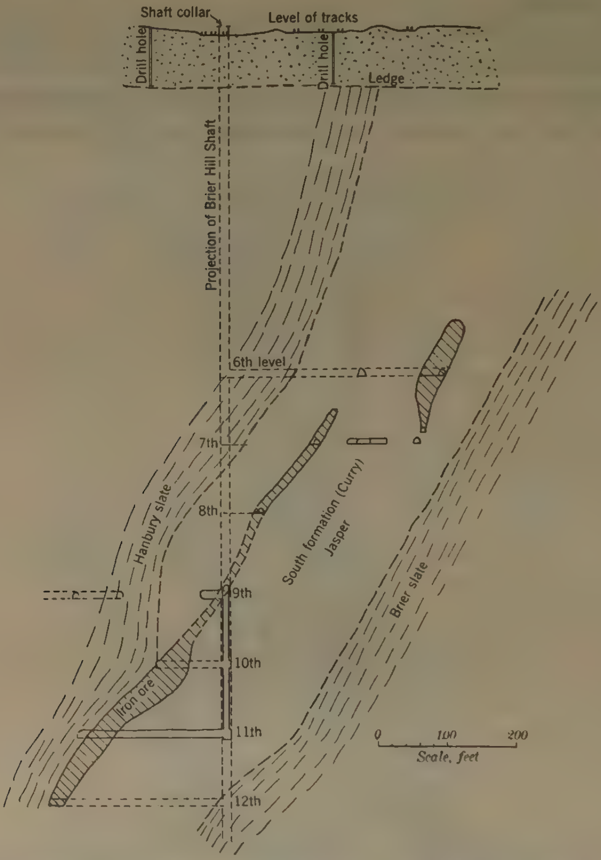

FIG. 12.—BRIER HILL MINE, CROSS-SECTION *J-J'*, 950 FT. WEST OF SHAFT. This section is close to the western terminus of the orebodies.

How high the caving had gone was not known at that time but it was thought that there was over 400 ft. of solid rock above the cave, which was confirmed later by crosscuts and test drill holes.

METHODS USED IN TESTING PROGRESS OF CAVING

The mine management adopted two of the methods suggested for testing the progress of caving, as follows:

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[2] For testing methods proposed see G. S. Rice and C. Enzian: Tests of Strength of Roof Supports Used in Anthracite Mines of Pennsylvania. U. S. Bur. Mines *Bull* 303 (1929). This proposed testing was not done.

1. Exploratory headings were driven from the fifth-level footwall through the jasper gangue with the intent of passing over the excavated area, to determine when caving or cracking had reached this elevation, and if it did possibly to hole into and fill with waste rock any cavity found.

2. Testing by boreholes from surface was carried on for several years and comprised the following: (1) Boring vertical diamond-drill holes from the surface on each side of the railroad tracks, where the conditions of underground caving were most acute; (2) boring several inclined holes, starting farther back from the tracks and crossing under them in the hanging wall above certain stopes; these holes to be cased through the drift formation, cemented to the ledge and then drilled into the rock; (3) testing the tightness of the rock from time to time by filling the holes with water, assuming that if the water did not run away rapidly cracks did not intersect the drill holes; (4) drilling the vertical holes until a cavity was run into, then recording its depth below the collar of the casing, and when the drill, lowered through the cavity, struck the top of the caved material in the stope, the difference in depth would give the height of the open cavity; (5) after making these measurements the hole would be plugged above where it entered the cavity and grouted for, say, 10 or 15 ft. above the plug. Water would then be run into the hole to find whether there were any large cracks above the plug; if only small cracks, the hole would be grouted to make it tight, or another plug would be inserted above; (6) to give immediate notice of caving that had broken rock at or above the concrete plug in the hole, a line of ½-in. pipe was to be inserted, the bottom resting on the cement and the top projecting 4 or 5 ft. above the top of the casing. A record of the height of this projection would be made so that in event of caving the distance of the descent of the small pipe could be measured. Then the pipe would be withdrawn and a new plug and cementing placed higher in the hole and the system of measuring by small pipe repeated.

The test drilling was started by a vertical hole begun Oct. 4, 1914, and finished Jan. 25, 1915. There were 17 holes: 10 vertical, 5 nearly vertical, and 2 highly inclined. The drilling was continued until February, 1918, and the observations carried on by the late F. A. Janson, mining engineer of the company.

Mr. Janson reported from time to time to Mr. Kelly, who furnished the writer with copies with his comments, on the progress of drilling and conditions shown in the mine. These are summarized as follows:

Up to Feb. 27, 1916, there had been taken out 50,000 tons of ore by the top-slicing and caving method used only in the extraction of pillars between the seventh and eighth levels in the area extending from 400 to 800 ft. west of the Brier Hill shaft. In all other parts of the mine the usual system of back stoping on square sets, with subsequent filling, was used, leaving very little unsupported ground open except drifts, crosscuts and rooms which were filled as required.

At that time (Feb. 27, 1916) Janson reported that the ore above the seventh level had all been taken out to a height about 60 ft. above the level, leaving a capping of rock 465 ft. high, to the top of the ledge. He stated that it was likely, and in fact probable, that the caving had extended considerably higher in places where the top slicing and caving system had been more extensively used, as for example, at a distance of 500 to 800 ft. west of the shaft, the largest pillars being at this point. That such was the case was indicated by the result of a diamond-drill hole put down 600 ft. west of the shaft.

The drill hole showed: surface (sand and gravel), 78 ft.; Hanbury slate, 349 ft.; jasper, 31½ ft.; total, 458½ ft. At a depth of 398 ft. a portion of the feed water was lost and at 427 ft. at the contact of the slates and the jasper, the entire amount of feed water disappeared, indicating that the disturbed ground had been reached. Drilling was continued in jasper to a depth of 458½ ft., when the rock became so broken as to make further progress impossible. It appeared at that time that the fracturing of the rock (not actual caved rock with vacant spaces) at this point had reached somewhere between 427 and 458 ft. below the surface, leaving a capping of undisturbed ground of approximately 340 feet.

Janson concluded at that time (Feb. 27, 1916) that there should be no danger of the caving reaching the surface, at least until the pillars had all been removed down to the ninth level, which would take at least a year and a half, unless the cave should take the form of a chimney.

Janson made another report May 6, 1916, stating that there had been a total extraction up to that time of 140,000 tons of ore by the caving method from pillars, leaving 60,000 tons to be extracted by that method down to the ninth level (800 ft. below surface). He further stated that ore on the tenth level was being taken out by the slicing and square-set method with filling, although at increased expense and remarked that the development work on the eleventh level indicated that it was near the bottom of the ore lens and, if this was the case, the ore would be exhausted in a short space of time and the danger to the railroad would be practically eliminated.

As regards the test drill holes, Janson reported (May 6, 1916) that three vertical holes and one inclined hole had been completed. To illustrate the method of testing, a partial record of these holes is given in Table 1.

SECOND REPORT OF SPERR AND RICE, OCT. 6, 1916

The changing situation caused by mining developments since the Commission's investigation in 1913 and indicated by the test drill holes,

TABLE 1.—*Partial Record of Drill Holes*

Drill Hole No.	Distance from Shaft, Ft.	Distance from Tracks, Ft.	Depth Drilled, Ft.	Surface Overburden, Ft.	Plugged, Ft.	Observations
1	500	101 N.	457	85	340	Showed no caved or broken ground. Water drained out in Feb., 1916. Hole still open for observation.
2	600	84 N.	458½	74	350	Lost feed water at 427 ft. Broken but not caved ground from 427 to 458½. Hole still holds water.
3	799	72 N.	502½	69	425	Showed no caved or disturbed ground. Hole still holds water.
4	600	265 S.	700	inclined N. 45°		Undercut tracks 250 ft. below surface. Hole still holds water.

led the railway and mine officials to have another investigation made by the same commission. This was done in October, 1916. Summarized extracts from the second report follow:

At the time of the previous inspection 10 levels were opened up, and in the lower levels the ore appeared to be much softer than it had been higher up. Hence it was thought necessary, on account of the danger to the workmen, to mine the ore by the caving method from the ninth level downward; but, since the drainage of water from this ore, it has developed strength so that it is now found practical to mine it by the filling method below the ninth level. The excavation already made by the caving method represented an unfilled opening of about 1,600,000 cu. ft., with approximately 400,000 cu. ft. yet to be taken out.

From the nature of the overlying Hanbury slates it was not thought likely that the unfilled space caused by the caving method would remain open; the unsupported slates probably would cave. A serious question was whether the slates would break in a mass and not in detail, by which latter process the cavity might become more or less filled by the falling pieces of rock.

AN EXPERIENCE IN CHIMNEY CAVING

An important experience of chimney caving bearing on this problem, (occurring since the report of the Commission in 1913) took place in workings of the West Vulcan shaft where an opening, from a heading driven in the Hanbury slate hanging wall, was made two sets wide and four sets long for the purpose of obtaining filling material. The caving was started in the opening, the material being drawn off through the chutes into tram cars. In about a year the caving had worked through to the surface in a vertical chimney having a cross-section through the rock about equal to the size of the initial opening at the tenth level. The dip of the slates was about 60° from the horizontal, but the cave cut diagonally through the bedding in the form of a vertical chimney. It was this experience that suggested the possibility of a similar occurrence in the mining operations under the railroad.

The danger in case a chimney was formed was that, on reaching the overlying sand and gravel, a rush of this material might be produced which would open a hole under or adjacent to the tracks so as to cause displacement of the rails and endanger trains.

The Commissioners found that, since the previous inspection, the eleventh level, which had been opened, showed a smaller thickness and length of orebody. Therefore, the indications were that the ore would probably not extend very much deeper unless some new orebody were discovered by subsequent development. Also, it was found that the ore between the tenth and eleventh levels was of such nature that from experience between the ninth and tenth it could probably be worked by the filling method. This again was favorable toward diminishing the danger that seemed imminent at the time of making the previous report.

Crosscuts and Drill Holes to Determine Extent of Caving

The mining company had spared no pains to obtain information relative to the condition of the ground over the area where the ore had been extracted by the caving method. . Three crosscuts were being driven on the fifth level (about 300 ft. below the rock ledge, see Figs. 1 and 5) to extend over the excavated area and under the railway tracks. This driving had not then been completed, but the first cross drift, which is 550 ft. west of the shaft, had extended through jasper into the hanging-wall slate but had not yet reached a point vertically over the caved area. It had encountered some shattered material and, at the time of the investigation, work had been temporarily suspended.

The Commission described the progress of the test drilling. The positions of the holes are shown on the map (Fig. 1) and in vertical sections (Figs. 2 to 15).

The character of the drift overburden disclosed by the drill holes was found to be of significance. The presence of hardpan and clay immediately over the rock ledge was shown and of thicknesses up to 33 ft.; only one hole showed as little as 4½ ft. In this hole, although over the clay, there was 7 ft. of sand, above that there was 18 ft. of hardpan and clay.

In reference to the clay above the ledge, as shown in all the holes, the Commission suggested that the presence of the clay over the ledge was distinctly favorable in preventing running of the sand or gravel through cracks, though it would not, of course, prevent a large opening from caving the sand and gravel above. Another favorable feature was the comparatively small quantity of water found remaining in the sands and gravel. Water is usually found in large amounts in glacial drift, but here it had probably been largely drained by the mines, also the surface of the ledge sloping to the south would tend to run off excess water.

SUMMARY OF CONCLUSIONS AND RECOMMENDATIONS IN COMMISSION'S REPORT OF 1916

"1. While the conditions are more favorable for mining by the filling method than at the time of the previous investigation in 1913, there are indications from the fracturing of the ground adjacent to the excavated orebody in the upper levels that caving is likely to continue upward, and it is possible that vertical chimneys may be started in the slate hanging wall. These caves or chimneys may not reach the surface, but this cannot be assured.

"2. That in view of the more limited extent of the orebody below the tenth level than was indicated by the cross-sections at that level in 1913, it does not seem necessary to move the tracks of the railway until some further important change is indicated:

"(*a*) By the possible finding of a large orebody or bodies in the lower levels and subsequent mining of them.

"(*b*) By the requirement of more extensive use of the caving method than is now under way or under contemplation.

"(*c*) By finding that the cavities or chimneys extend within dangerous proximity of the overlying alluvium or that such caving is proceeding upward at so rapid a rate that the surface will be reached in a short time.

"That the present plan of drilling boreholes from the surface into ground overlying the orebodies and adjacent to the railroad be continued, the vertical holes being placed not over 50 ft. apart on either side of the railroad tracks.

"That certain of these holes over the largest areas of caved ground be drilled to the cavity and that measurements be made to determine the space between the top of the cavity and the top of the broken material below. The smaller this space is found to be, the less will be the danger of a sudden rush to the surface.

" . . . That crosscuts on the fifth level be extended southward across the contact and into the hanging-wall slate to a point 25 ft. south of the railway tracks, or, if the conditions due to fracture of the ground make it appear dangerous to continue such drifts, then diamond-drill holes be bored ahead in a vertical fan shape to determine where the top of the cavity now is.

"That if by any of the above means it is indicated that caving or chimneying is proceeding so rapidly upward as possibly to reach the top of the rock at the bottom of the alluvium within a year, an effort be made hydraulically to fill the cavities through large boreholes with sand and gravel by the well-known methods used in the Pennsylvania anthracite district and European mines, particularly in Upper Silesia. It is not proposed that this method be employed unless dangerous conditions develop, as it would complicate the present mining through introduction of large quantities of water into the mine."[3]

Progress Reports of Kelly and Janson

Subsequent to the foregoing Commission report, Janson at intervals made informal progress reports, abstracts of which follow:

On Jan. 24, 1917, Janson, after describing detailed progress in drilling test holes to cavities, one of which showed a vacancy over the fallen rock of about 3 ft., reported these underground observations:

"On the fifth level, about 500 ft. west of the shaft (Brier Hill) a crosscut has been extended south through jasper into the hanging-wall slates. At a point near No. 1 hole it encountered ground somewhat shattered, and further work was considered unsafe.

"About 200 ft. farther west on the same level another crosscut had been extended south through the jasper and into the hanging-wall slates to within 50 ft. of a point directly below the main-line tracks. This heading did not encounter any caved or shattered ground.

"It was the intention to continue, but the miners thought that they had heard some sounds of falling ground and became frightened. * * * Rice and Sperr recommended, if such an emergency should occur, drilling of some horizontal fan-shaped holes extending under the tracks from one of the crosscut headings. I would propose that this be done as soon as we are through drilling on the surface."

Janson, in a report May 17, 1917, gave a general description of the drill-hole development and tests. Referring to his report of Jan. 24, he said that there had been further settlement of observation pipes in holes 5 and 6 since then and he decided to raise the pipes and to plug the holes to establish new points of observation higher up "in accordance with suggestions made by Rice in his letter to you (Kelly) of Feb. 17, 1917."

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[3] Since 1913, (beginning about 1927), filling material transported by compressed air has been largely substituted in German mines for hydraulic stowing. This system of stowing by compressed air might be an admirable means of filling cavities in future cases similar to that at the Brier Hill mine.

Hole 10, at 700 ft. west of the shaft and 20½ ft. south of the main-line tracks, encountered a small crack at 428½ ft., another at 453 ft., and still another at 441 ft. At 526½ ft. caved ground was encountered.

Hole 11, at 500 ft. west of the shaft and 32 ft. north of the main-line tracks, encountered at depths of 273½ to 331 ft. a series of small cracks. Drilling was continued to 387 ft., after an intermission in the drilling because a movement cut off the hole at 273½ ft. from the surface. Further investigation indicated no arching or open spaces, then a plug was put in at 260.75 ft. on which an observation pipe rested.

May 18, 1917, Kelly, in transmitting the foregoing report to W. H. Finley, of the Chicago and North Western Railroad, commented:

" * * * The developments since the last report indicate the cracking of the rock is progressing upward. * * * While the rate of progress of the caving observed during the last four months would indicate that it might be a year before the cave reaches the top of the rock, we have no assurance whatever that the observed rate of cracking will continue the same. * * * I feel it necessary to advise that the speed of the passenger trains and probably also of the freight trains be reduced when passing over the spot, to a low rate, say about 10 miles an hour."

Evidently this opinion caused the railway officials such anxiety over the possibility of a train accident that they decided to move the tracks without waiting for more definite indications of caving reaching the surface, and the tracks were moved in 1916.

In November, 1921, the writer visited the Brier Hill mine and inspected the workings under the former location of the railroad tracks, as far as these were accessible, and on the surface observed the tops of the various drill holes.

At that time, two years after the tracks had been moved, there was still no indication of surface subsidence, although the measuring devices —the observation pipes—in the drill holes showed progressive breaking of the rock proceeding upward toward the ledge.

Although the practical need of the investigation had passed with the moving of the railroad tracks, in view of the unusual record of progressive ground movement with definite measurements it seemed highly desirable to make a permanent record of the successive stages of the movement with relation to the mining excavations and the methods of obtaining the measurements. Accordingly, the writer proposed to Kelly that, with his cooperation and the assistance of Janson a paper recording this valuable study be prepared.

This proposal was cordially agreed to and the data plans and cross-sections were assembled.

Mr. Kelly, in a letter dated Feb. 6, 1922, stated: "In accordance with the memorandum that you left with us for data upon the ground movement at the Brier Hill mine, I enclose statements and blueprints (maps and sections[4] showing what appears from our records, together with a statement by W. Siebenthal (assistant mining engineer). Janson has looked over these papers and blueprints and is satisfied that they are as complete as can be made. . "

Janson died soon after he and Siebenthal had assembled the drawings and data requested.

At the time (February, 1922) this material was assembled it was expected that a paper would be prepared for immediate publication, but later it was decided that it would be advisable to wait until the surface caved, which it was then thought might occur in the near future. Siebenthal's statement contained the following information:

" . . . Practically no exterior filling has been introduced into the openings from which ore has been extracted other than that run from old workings and from new drifting, since October, 1916, the date of the report of Rice and Sperr.

"Caving ground and old filling have, we believe, largely replaced the ore mined (i.e., by filling the excavations with broken rock).

"During the period 1916–21, inclusive, there has been mined and hoisted from the Brier Hill mine, 670,912 tons of ore, which would represent an excavation of approximately 7,700,000 cu. ft. Our ore report of December 31, 1921 shows a tonnage of 116,450 tons (approximately 1,164,000 cu. ft.) in the mine above the twelfth or bottom level.

"Our observations do not indicate any considerable 'drop' of ground (i.e., rock, not surface) at any time in the caving process, but rather a cracking and settling of the undermined area over the filling as the same would settle and shrink from decaying timber and compacting pressure * * * . We submit a longitudinal section through each of the three lines of drill holes, with graphs representing approximately the position of the broken ground on Dec. 31 of each 1917, 1918, 1919, and 1920 and the last made on July 30, 1921." (See Figs. 13, 14, and 15.)

Following submission of these data in 1922, letters were interchanged at intervals between the writer and the mine officials. Jan. 3, 1923, Mr. Kelly sent some data on measurements in the drill holes and stated, "There has been no caving of the surface, but the excavation of ore without filling is going on, so it is impossible to say that the experiment has come to an end." Later in that year Kelly retired from the mine management but kept in touch with the Brier Hill situation.

EVENTS SUBSEQUENT TO 1923

In October, 1926, the writer again visited the Brier Hill mine. At that time observation pipes in eight of the holes were still projecting.

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[4] From which Figs. 1 to 15 have been redrawn for this publication by engineers of the Bureau of Mines, without essential change except for adding reference lines.

Mr. Denton, mining engineer of the company, measured the projection height above ground, Oct. 15, 1926 (memorandum dated Jan. 3, 1927, by D. B. Cavan, superintendent, to William Kelly). Further underground caving since May, 1922 was shown in these holes: No. 3, 1.64 ft. drop at bottom of hole; No. 7, 2.51 ft.; No. 8, 0.13 ft.; and No. 10, 2.85 ft.[3]

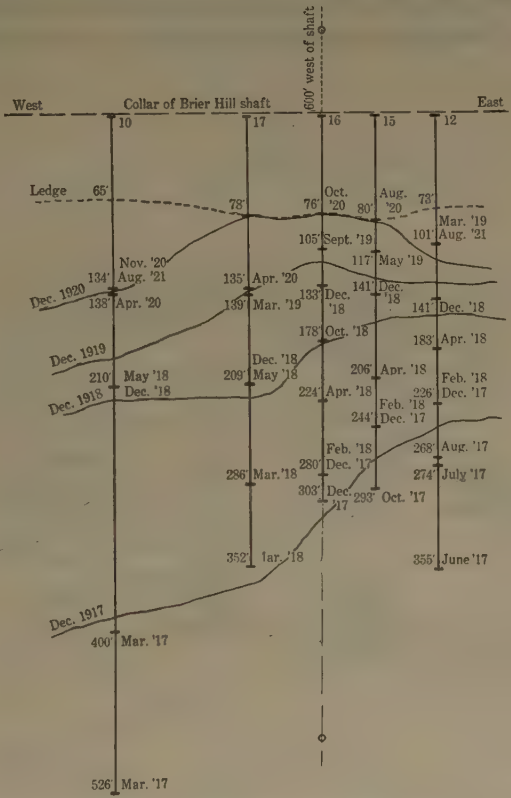

FIG. 13.—LONGITUDINAL SECTION, THROUGH SOUTHERLY LINE OF HOLES 10, 17, 16, 15, 12 (WEST TO EAST).

The other drill holes did not indicate that movement had taken place since 1922. The writer of this paper noted at the time of his visit (October, 1926), that the surface of the old railroad yard, with tracks removed, gave no indication, observable to the eye, of either caving or subsidence.

W. P. Chinn, general manager of the mining department of Pickands, Mather & Co., which now manages the Penn Iron Mining Co. property, in a recent letter (Aug. 15, 1933) states: "up to the present no break in the surface has occurred."

SUMMARY AND CONCLUSIONS

It appears likely from the drill-hole records that sometime there may be a break to the surface, but more probably a slight general subsidence of the surface, although it was thought by some that caving might take place in 1917. It seems advisable to wait no longer, however, in presenting the valuable data for permanent record. Ground movements and effects have so many physical variables it is not possible to treat mathematically, so the best that can be done is to record as precisely as possible observed facts including geological and mining conditions.

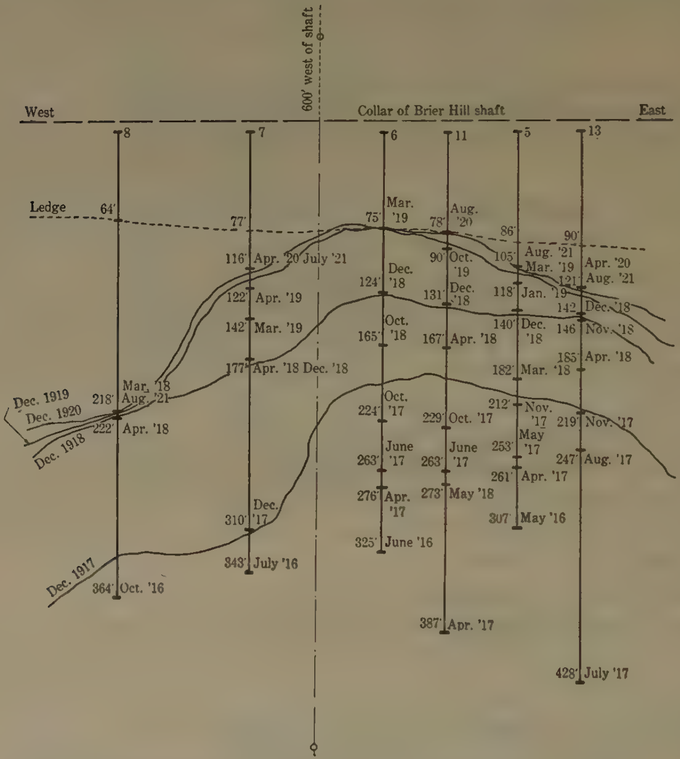

FIG. 14.—LONGITUDINAL SECTION, THROUGH MIDDLE LINE OF HOLES 8, 7, 6, 11, 5, 13 (WEST TO EAST).

In this summary it is not necessary to give the detailed records of the ground movement at the Brier Hill mine. The principal drill-hole findings are shown in condensed form in the longitudinal diagrams in Figs. 13, 14, and 15; the cross-sections (Figs. 3 to 11), longitudinal section (Fig. 2) and plan view (Fig. 1) of the affected parts of the mine show other details graphically. The important features of value for possible application in cases occurring elsewhere of ground movement with possible surface subsidence, from mining in steeply pitching beds or veins, are summarized as follows:

How Ground May Break over Mining Excavations

From the Brier Hill experience we learn that in mining a vein or bed with a dip of over 65°, caving may not keep within the formation of the walls, but as in this instance go more or less vertically upward through overlying bedded rock, the planes of rupture diagonally crossing the bedding planes. Another example previously cited was that which occurred in a distant part of the mine where a "chimney" formed in drawing off material from the hanging wall to be used in filling elsewhere.

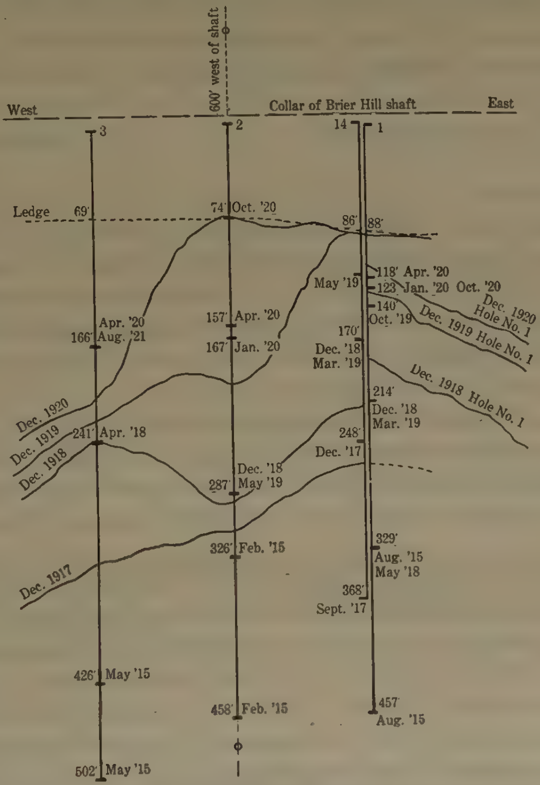

FIG. 15.—LONGITUDINAL SECTION, THROUGH NORTHERLY LINE OF HOLES 3, 2, 14, 1 (WEST TO EAST).

Another condition may arise where parallel fault planes cut across a formation so that side support is not given. A remarkable case of this occurred at the Athens iron mine, Michigan, where a block of solid jasper, 2000 ft. deep between nearly vertical planes, subsided in June, 1932, over a mined depth of ore bed of only 200 ft.[5]

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[5] C. W. Allen: Subsidence Resulting from the Athens System of Mining at Negaunee, Michigan. See page 195, this volume.

At Brier Hill there were no major faults crossing the formation and chimneying did not occur under the tracks as indicated by the data obtained by the test drill holes, but instead, as is more usual, continued spalling of rock from the underside of arch cavities occurred, gradually filling the cavity through occupying more volume than the original rock as the progressive upward breaking occurred.

Under normal conditions in a pitching bed, evidently the horizontal width and depth of a mine cavity, the amount of prior filling brought in from outside, and the thickness of the capping rocks are interrelated factors as to the probability of caving going to the surface.

Whether or not breaking will go through the hanging wall in a steep pitching bed or vein appears to depend both on the strength of the hanging wall and the amount of support given by remaining pillars and filling material.

It is not meant to suggest in this discussion that there is what has been termed fallaciously a "harmless depth of mining." Where it relates to workings of great lateral width in a flat-lying bed, subsidence of the surface will inevitably occur, the magnitude of subsidence varying with the depth and whether or not "back-filling" material has been introduced into the excavation. If the excavation is tightly packed, especially if done by hydraulic stowing with clean sand and gravel, the surface subsidence may be very little and not damaging to surface improvements, as demonstrated by numerous instances at French and German mines where hydraulic stowing is extensively used in workings under towns and canals. Dry filling is less effective in getting tight packing.

Estimate of Volume of Ore Removed by Caving at Brier Hill Mine

Important considerations in studying the possibility of subsidence resulting from mining a more or less irregular orebody, such as the Brier Hill main lens, are the quantity of ore to be mined under a specific area and its depth below the rock ledge, and the amount of filling done. In this case it is difficult to obtain precise figures as to the specific area. The tonnages given in Table 2 were reported by the mine management, of ore hoisted from the areas after the caving system had been found necessary by the mining conditions, including that estimated to be remaining at date of estimate:

TABLE 2.—*Tonnages of Ore Hoisted and to be Removed by Caving, Dec. 31, 1921*

	TONS OF 2240 LB.
Between seventh and ninth levels (Janson's letter, May 16, 1916)	140,000
Ore mined between 1916 and 1921 inclusive	670,912
Ore hoisted and remaining to be mined to ninth level (Siebenthal's estimate Dec. 31, 1921)	60,000
Total	870,912

Although the ore below the ninth level was reported to be mined by slicing and filling, for purposes of estimating the volume removed, we will allow to provide a margin of safety, that one million tons of ore (of 2240 lb.) was ultimately mined from the area under consideration. Taking the weight of hematite at 312 lb. per cu. ft. there would be 7.2 cu. ft. per ton; and on these assumptions 7,200,000 cu. ft. of space would result.

Estimating from the sections and maps, the length of lens involved in possible subsidence is about 500 ft., the vertical height 571 ft. and the average horizontal width approximately 25 ft. These figures give the approximate volume of space occupied by the ore as above estimated, 7,200,000 cu. ft.

If it is assumed that this space is ultimately filled by broken jasper and slate from the hanging wall, and this broken material recompresses under its own load to a volume of 1½ times its volume in place, and the breakage proceeds upwards with vertical planes 25 ft. apart, it would require a column of solid rock 457 ft. high to fill, when broken, 571 ft. vertical height of lens excavation, and an additional column of 366 ft. to fill the caved space above the top of the original ore lens, making a total vertical height of 937 ft. to which breaking would go under the assumptions made.

The average height from the bottom of the lens to the rock ledge is 952 ft., giving a margin, under the assumptions, of only 15 ft. of solid rock up to the drift overburden. However, it is not at all probable that the breakage of the rock overlying the mining excavation would be confined within the assumed limiting vertical planes only 25 ft. apart, but would extend on either side. The horizontal projection of the lens in its most central section is about 300 ft. wide and all of the ground vertically above, and probably more, might contribute broken rock from the hanging wall to fill the vacancy.

Unless the broken rock from spalling off, plus filling material brought in, is compressed to less than 1¼ times its original volume the vacancy over the broken filling might be reduced to practically nothing when the arching reached an elevation much less than 952 ft. above the bottom of the ore lens. However, cracking of the rock capping did undoubtedly occur extending up to the ledge as indicated by the drill holes.

While the foregoing reasoning is based on assumptions, it may explain why subsidence of surface has not occurred since mining ceased in the particular area in 1923. Whether or not in future years the broken rock filling will further compress or the spaces between fragments gradually fill by precipitation of solids from percolating water is conjectural.

Where a bed of hardpan or clay of some thickness covers a rock ledge over a mined area, as at Brier Hill, its presence is of great importance in preventing fine sands containing more or less water from running into cracks. This was evidently a factor of safety not fully known at the beginning of the investigation, and another was that excess water had drained off, partly due to the slope of the ledge and partly by flow to the neighboring shafts. It points to the advisability, in other instances of mining below sand or gravel beds saturated with water, of putting down adjacent drainage wells and pumping out the water where there is danger of breaks from mining extending up to the water-bearing ground.

Merits of System of Testing Ground Movement by Drill Holes

The system of drilling test holes to determine the presence of cavities and the height of the open space over the filling has great value where surface structures are threatened by mining. The value is greatly enhanced if, as at Brier Hill, the drill holes, after being cased through the drift overburden and cemented to the ledge, are subsequently employed to test the tightness of the rock. As previously described, this is done by filling the holes with water, and when a cave or crack is subsequently indicated in a hole by sudden loss of water, after recording the measurements, putting in a plug with cement above the cave or crack. Then an observation rod, made of a string of small coupled pipes, is placed in the hole resting loosely on the plug, the top of which projects above the drill-hole casing. By this means measurements can be easily made at stated intervals, to find when a break or cave has occurred and, after measuring the drop, repeating this procedure of putting in a plug higher up. Thus the rate of caving and cracking upward is obtained and warning given of an impending danger of surface subsidence.

ACKNOWLEDGMENTS

In closing, the writer makes acknowledgment to William Kelly for making the investigation possible and for the information furnished, and expresses appreciation of the work of the late F. A. Janson, who recorded most of the data obtained, and of the late Prof. F. W. Sperr, who with the writer prepared the Commission's reports of 1913 and 1916.

DISCUSSION

(*H. G. Moulton presiding*)

S. ROYCE,* Crystal Falls, Mich.—What is your conclusion as to what may be expected in that immediate area in the future? Do you feel that the equation is somewhere nearly satisfied by the natural filling that has occurred plus the artificial? We have as yet absolutely no sign of subsidence over this area, at surface.

G. S. RICE.—I am at sea about it. In the beginning of the investigation, I thought that there was no immediate hazard. The railroad company and Mr. Kelly of the mining company thought that there was, and Mr. Kelly wrote to me recently that he still thinks that some day it will cave. I think that the question is in the balance on it.

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* Mining Engineer and Geologist.

We do not know to what extent the broken rocks will recompress nor how much infiltration will do toward filling the voids. Manifestly the rocks up to the clays and sands are cracked; that is shown by the drill holes but we do not know whether the surface of the rock ledge has sunk appreciably. You know that region, and how far infiltration of water, which is continually moving over the surface of planed-off concealed rock outcrops, is going to carry down substances through the cracks in the rock filling material to fill the voids.

S. Royce.—You mean in solution?

G. S. Rice.—Yes, either in solution or suspension.

S. Royce.—We have no evidence of anything important in the way of consolidation by infiltration anywhere about the mine workings in our area, or around old rock piles, or at any of the places where it might be expected. There is a material in the region known as tillite, which is the partly consolidated base of the glacial drift. It is rather hard to tell from the Cambrian sandstone in occasional places. So something of the sort may have happened in the period between the glacier and now, but that does not offer us much hope for anything happening in our immediate human concern with that property.

G. S. Rice.—When we began this inquiry, as I said in the paper, it was thought that the gravels and sands might lie directly on top of the rock, and that if the cave-ins occurred like those at some of the mines in the neighborhood, the sands would follow down immediately, but the fact that in the first drilling we found a layer of stiff clay lying immediately over the ledge altered the situation considerably as to fear of immediate danger to the railroad tracks from water suddenly rushing in and creating a cavity extending to the surface—at least it altered it in my mind.

There was another thing that was discovered by the drilling. The sands and gravels that held water originally had been more or less drained to the south over the sloping ledge and also by the existing mines. That also was indicated subsequently in sinking a water shaft. Thus we were not dealing with a very wet formation, which was one of the things feared in 1913 and to a lesser extent up to 1916.

As a matter of obtaining scientific information, I had hoped that those tracks would be left undisturbed until we had found out more precisely what was liable to happen. If there had been obtained only a few more records by drilling and testing the recompression of broken slates, it would be very interesting from a scientific point of view.

S. Royce.—We are now having some caving on the East Vulcan mine. That, again, is working almost vertically. It is working right across the formation from near the footwall. You are dealing only with the hanging members of the formation in the area considered in your paper. The ore that is causing the East Vulcan caving is down near the footwall. Caving has worked nearly vertically across the Brier slate, and comes to surface at an angle of about 80° from horizontal. Have you any data from which you would want to say what the average angle of slope of the ultimate rest of caves is apt to be in various rocks, free from fault planes?

G. S. Rice.—The angle depends, I think, upon the strength of the rocks and the pitch of the formation and extent of the excavation.

S. Royce.—The usual assumption is that the angle of slope is not far from 70°, say 69°.

G. S. Rice.—That may be so where you are dealing with a large excavation that is fairly deep, say 500 ft. or more with overlying homogeneous rock, but where there are inclined bedding planes or strongly marked joints that form planes for slippage the angles may be very flat.

S. Royce.—You feel that that is fairly widespread experience?

G. S. Rice.—Yes, but I say it with a good deal of hesitation without knowing the conditions, because of the experience of movement on slippage planes at Bisbee and other mining districts. The great Turtle Mountain slide, Alberta, in 1903, thought to have been induced by coal mining in an outcrop at the foot of the mountain, was found to have been on joint planes of the limestone. These joint planes inclined about 40° toward the valley.[6] Here there was no lateral support, which is usually found in mining. The angle of repose of the coarse rock debris was 30° to 32°.

S. Royce.—There has been much subsidence study on the Gogebic in recent years, and we have gotten the half to one slope in the footwall, but a considerably flatter slope across into the hanging.

In collecting data on how caving has acted in this immediate old Menominee Range area, the realized slopes so far, we have not been able to find any that ran flatter than the quarter to one, and some as steep as a tenth to one. Those are mines of considerable age, and the bulk of the workings have had no registration at surface at all.

G. S. Rice.—Yet a case was reported at last year's meeting of chimney caving from deep mining.

S. Royce.—But that was nearly vertical, I take it.

G. S. Rice.—Yes. It occurred at the Athens iron mine, Michigan, and is described in a paper by Mr. Allen.[7] It is a very remarkable instance of a block bounded by more or less vertical fault planes, and not due to successive spalling off over the mining excavation, as in the Brier Hill mine.

R. D. Hall,* New York, N. Y.—I think some investigation should be made as to the chemical character of the rock materials of the mine, and as to the changes that take place in them by their reaction with solutions. Some rocks expand considerably from chemical reactions and assist greatly in filling vacancies.

G. S. Rice.—An interesting thought.

S. Royce.—In the pre-Cambrian iron formation under discussion there is comparatively little opportunity for chemical alteration after mining. That would not apply to many areas, of course.

R. D. Hall.—Perhaps there are chemicals that produce sulfates and some feldspar on which sulfates may react.

S. Royce.—Not much of either. Feldspar is almost absent from the iron formation, as you know, and sulfur never can exceed 0.1 per cent or the mine would not be operating in that area.

H. Briggs,† Edinburgh, Scotland (written discussion).—The remarkable series of observations carried out over so long a period by the author and his associates cannot fail to have a profound influence on views concerning subsidence over workings in steeply inclined deposits. The new technique they introduced for ascertaining the rate of the upward extension of ground movement is one of the most outstanding of those that have of late years been applied to the elucidation of subsidence problems, and, with one addition, it is likely to be copied by other investigators.

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[6] Report of Commission to Investigate Turtle Mountain Frank, Alberta, 1911. Canadian Dept. of Mines *Mem.* 27.

[7] See page 195 this volume.

* Engineering Editor, *Coal Age*.

† Professor, University of Edinburgh.

The addition to which I refer is the leveling of the surface at intervals during the period of observation. Leveling does not appear to have been carried out at Brier Hill mine; when it came to the question of surface settlement, the observers were guided, I judge, by visual indications only. Yet beds bend before they break; they may, indeed, bend without breaking. The dome of disturbance reached the ledge about 1919 to 1920; but before then there may have been some sagging of the slate beds, and, if so, the surface would be affected. I understand that the hanging wall is strong and that the amplitude of bending, if present, is therefore likely to be small; yet in softer ground any settlement of the upper part of a borehole due to the bending of the beds below would tend to mask the indications the hole is intended to provide.

I always feel considerable doubt as to the validity of calculations based on the assumption that the space underground is filled up with broken rock falling in from above, the broken rock being assumed to occupy, say, 1½ times its original volume. Calculations of this kind led to the fallacious doctrine of a "harmless depth." The fact is that, whatever the system of mining a tubular or lenticular deposit may be, the tunneling down into the excavations of a broken, rubbly mass, if it occurs at all, is confined to the lowest parts of the roof or hanging wall. Above that, when the strata subside they do so along planes of shear that allow the movement to occur as a series of steps. This adjustment of position has, in instances I have examined, taken place with only a light distention of the mass. The breaks along an extending series of dome-shaped surfaces which the author records would, I suggest, close up again with but a negligible enlargement of volume. Using data given by Herbert and Rutledge[8] bearing on longwall (coal) workings in Illinois, I calculated that the subsidence of the beds involved an enlargement of their total thickness by only 0.13 per cent. This increase is so slight that nothing in the nature of a smashing up of the strata above the seam—or even of one thick stratum—can have occurred.[9]

Directed as they are to the solution of a specific problem, the results set forth in the paper do not pretend to furnish a complete record of events. The story is unfinished; and though subsequent movements may be small, it is safe to predict that, before static conditions are completely restored, there will be a "draw" or lateral extension of the movement away from the vertical plane passing through the Brier Hill shaft. It would be most interesting to have Mr. Rice's views on the probable width of draw. Having regard to the valuable observations of W. R. Crane in Michigan,[10] does he expect "dip-slipping" to take place on the rise side, eventually involving an angle of draw on that side coinciding with the dip of the beds?

Judging from steep-seam mining in this country a still larger draw is at long last to be expected on the dip side of the workings. Within three miles of me as I write are collieries operating on seams of every inclination from zero to 85°. Inclinations about that of the ore lens at Brier Hill are common on the west side of this coal field of Midlothian, and some knowledge has been gained of the draw towards the dip. The astonishing thing is its great expanse in that direction, for the draw sometimes reaches almost as far as a line drawn at right angles to the bedding from the lower edge of the mined area.

To my mind, by far the most intriguing of all the observations recorded in Mr. Rice's paper is that headed "An Experience in Chimney Caving." That so small an underground opening should affect the surface at all is remarkable enough; but that it should do so through the medium of a parallel-sided "chimney" going through straight up without heeding the stratification is amazing. May I ask for the depth and exact dimensions of the excavation, and whether the superadjacent ground had been broken by previous subsidence arising from operations in the ore lens? Are the joints of the slate (or the cleavage planes) vertical, and can they have had an influence resembling that of the nearly vertical faults at the Athens mine, Michigan? And, again, what was the amplitude of the movement at the top of the "chimney"?

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[8] Subsidence Due to Coal Mining in Illinois. U. S. Bur. Mines *Bull.* 237 (1927).

[9] H. Briggs: Mining Subsidence, 137. London, 1929.

[10] Subsidence and Ground Movement in the Copper and Iron Mines of the Upper Peninsula, Michigan. U. S. Bur. Mines *Bull.* 295 (1929).

We are all aware that when a small opening is made in strong ground, the roof falls sooner or later and goes on falling until it has made for itself a Gothic arch; after which, if the supports of the arch are "competent" (to use Bailey Willis' phraseology) nothing further happens. We are equally familiar with the outwardly inclined lines of break that result from the extraction of a deposit over a wide area. If, then, the lines of break incline inwards over narrow and outwards over wide excavations, it may be allowed as a theoretical possibility that there is a certain width of excavation for which the breaks go vertically upwards. Until I read Mr. Rice's paper, however, I have to admit that this argument seemed too specious and flimsy to be worth even a passing notice: I never thought that what appeared to be so unlikely a chance would ever "come off." The peculiar fascination of subsidence is that the unexpected always happens: it is always ready to offer a new deal.

Mr. Rice shares my interest in the natural pipes of the Mons coal field, Belgium. These are huge perforations through the coal measures, extending from unknown depths to the old Wealden land surface, and filled with Coal Measure debris and Mesozoic material. None of the theories of formation is satisfactory. In his latest study of this, perhaps the most baffling of all geological phenomena, Prof. X. Stainier demolishes each hypothesis in turn and then finds himself unable to put anything in their place. One of these hypotheses is that the pipes may be due to subsidence into cavernous openings in the Carboniferous limestone lying below the coal measures. After reading of the "chimney" at Brier Hill that explanation of the Mons pipes becomes appreciably less fantastic.

G. S. Rice.—Replying to the comment made by Professor Briggs about the desirability of leveling to determine the precise amount of subsidence if any occurs, the leveling of the surface points was carried on by the late Mr. Janson, who was an extremely careful engineer. In the condensation of the large mass of data regarding the Brier Hill case, it was supposed that it would be assumed by the reader that careful leveling determinations had been made, as has been done in most other papers submitted to the Ground Movement and Subsidence Committee.

Mr. Janson used the collar of the Brier Hill concrete-lined shaft for his reference datum, as is diagrammatically indicated in each of the sections Figs. 3 to 15. Although figures for the elevations of the tops of the drill holes and of the underground mine workings are not given in the condensed redrawn sections, the originals were very carefully platted by Mr. Janson with reference to relative elevation. In his detailed reports he indicated the depths of the various elevations to one-hundredth of a foot.

When in 1926 I again took up with Mr. William Kelly, the former general manager of the mine, the final preparation of a paper, I asked for data about the elevations of the drill holes at that time. My request was referred to the then superintendent of the mine, Mr. D. B. Cavan, and he reported that he had Mr. Denton, the engineer, search for the old notebooks to locate the elevations of the drill holes but had not found them, so he took a new set of elevations Oct. 15, 1926, starting at the Brier Hill shaft collar and determining the elevations of the different drill holes with respect to it, to one-hundredth of a foot. He stated in his report that "these new elevations when platted on the old profiles showed no change as regards surface subsidence."

While in the interest of precision in such matters, it would have been desirable to report the exact figures for the elevations of the tops of the drill casings serving as monuments, it was less important from a practical standpoint than in many other instances, inasmuch as if any subsidence did occur it was likely to be of magnitude.

As I explained briefly in the paper, where the caving system has been followed in the Michigan iron mines, this usually has extended to the surface and caused large surface caves, and such was the case in another part of the same property as mentioned on page 127.

In reply to Professor Briggs' statement that there is doubt as to the validity of calculations based on the assumption that the space underground is filled with broken rock falling in from above, as discussed in the paper, I agree with him that the conditions where this may be relied upon for predicting the amount of subsidence are unusual. This has been especially true in the Michigan iron range and that is what makes this case especially interesting and valuable to record. It is hoped that in the future, leveling of the surface monuments will be done at Brier Hill when any indications of subsidence are suspected and that the results will be reported to the Institute.

I do not think that Professor Briggs should compare this case with that studied by Messrs. Herbert and Rutledge, made under my direction, on subsidence from flat-lying longwall workings in Illinois. In the latter case, an extensive area of a bed only 3 ft. thick had been mined out completely by longwall in a systematic manner, so it would be expected that the overlying clay shales would in time be compressed to their original density. In fact, only the lower layers of the immediate shale roof would be broken. I have observed that, in driving headings in that field through the old goave, after a lapse of years the reconsolidated broken shale appeared to be as hard and dense as in the original undisturbed shale. On the other hand, the slates and jasper formation at Brier Hill are very much harder rocks, which break in angular fragments.

Professor Briggs asks about the magnitude of the movement in the block cave at the Athens mine. I can only quote from the paper by Mr. Allen, that the ore mined from the block "represented an excavation roughly 250 ft. deep and 250 ft. wide, the center of which was 2100 ft. below surface." But there was an adjacent block over which a surface cave first appeared (June 19, 1932). The detailed measurements of the surface cave which appeared in the overlying sands and gravel are not given in Mr. Allen's paper, but presumably the surface subsidence is continuing.

The following interesting statement is made by Mr. Allen (p. 199): "The subsidence of a block of jasper capping nearly 1900 ft. thick resulted from a vertical opening of about one-tenth that height; convincing proof that *subsidence theories must largely take into account existing geological conditions.*"

As concerns the questions raised by Professor Briggs as to "an experience in chimney caving," referred to briefly in this paper, I will now abstract more fully from the second report of the Brier Hill Commission (Oct. 6, 1916):

"An important experience may be cited of an occurrence above the 10th level of the West Vulcan shaft (the workings of which adjoined the Brier Hill mine on the east) at which level an opening was made in the Hanbury slates two sets wide and four sets long (requiring an excavation of about 14 ft. wide and 28 ft. long) for the purpose of obtaining filling material and which was expected to cave in to the area thus opened. The caving proceeded according to expectations, the material being drawn off through chutes into tram cars and dumped into rooms which had been mined out from lower levels. In about a year's time the cave had worked through to the surface in a vertical chimney having a cross-section about equal to the size of the initial opening at the 10th level. The dip of the slates was about 60° from horizontal but the cave cut diagonally through the bedding in the form of a vertical chimney."

The results were remarkable, not only that the natural stoping was apparently not affected by the inclined bedding planes of the slates but also in the speed of the self-stoping up through about 900 ft. of slate to the overlying glacial drift in a year's time.

This information was given by the mine management to the commission; these workings were closed at the time and only the conical pit at the surface was observable.

The bottom of the surface pit was then (1916) about 30 or 40 ft. deep, and the chimney in the slates was closed. Probably the sides of the chimney through the slates had sloughed off and the surface drift had run in over the broken slate filling.

The existence of fault planes, such as described by Allen in the block caving at the Athens mine, were not reported. As regards the effect of prior mining of the ore lenses in the Vulcan mine above the tenth level, these lenses lay to the north of the chimney and the raise angle extended away from it and there was no indication that this prior mining affected the slates. It is remotely possible, however, that the cross-joints of the slate might have been slightly opened by a swing of the hanging wall of proportions not noticeable to the eye.

In reference to the "natural pipes of Mons," Belgium, which are more or less smooth circular or elliptical holes through the coal measures, filled with later marine sediments containing many fossils, it was the chimney stoping at the West Vulcan mine that suggested to me that these pipes might be formed in a similar manner. That is, assuming there were great solution cavities in limestone below the coal measures and natural stoping occurred. I further conjectured that when the coal measures were being folded upward through the overlying sea in Tertiary times, the sides of the holes were smoothed by great waterfalls; then with further rise of the coal measures rocks, the initial Tertiary marine deposits filled the holes and subsequent Tertiary deposits spread unconformably over the tilted coal measures.

Similar instances of natural stoping appears to have taken place in Upper Cretaceous beds of southeastern Colorado in the vicinity of the Arkansas River, recently described as fossil sink-holes[11] by Messrs. Dane and Pierce, of the U. S. Geological Survey.

S. Royce (written discussion).—The limit of subsidence in the footwall on the Gogebic Range is very greatly steeper than the limit of subsidence in the hanging wall. The subsidence is so nearly parallel to the footwall that subsidence originating from depth of 2000 ft. at the footwall contact only reached some 200 ft. into the footwall itself at surface, the dip of the footwall averaging about 68° in the Newport area. This means that actual angle of the boundary of subsidence is not far from 65°. Toward the hanging the inclination of the boundary of subsidence is more like 40° or 45°. These figures are not given as exact figures but are approximate and from familiarity with the situation.

Ultimate subsidence slopes on the Eastern Menominee Range are very much steeper. Instead of being ½ to 1, the flattest that we have found is about ⅕ to 1, and the initial break on a recent one is about ⅒ to 1 toward the hanging wall with no registering toward the footwall as yet.

This would seem to indicate that slopes toward the hanging are in both cases flatter than toward the foot, and what Mr. Rice says as to the Gogebic agrees with my own observation, which is that the bedding planes are the dominant factor in determining the south limit of the caving, and the joint planes or possibly dike planes (which, of course, coincide with joint planes) are the dominant factor in determining the pull toward the hanging.

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[11] C. H. Dane and W. G. Pierce: Fossil Sink-holes in Cretaceous Beds of Prowers County, Colorado. *Jnl.* Wash. Acad. Sci. (1934) **24,** 194.

Referring to the discussion as to whether any subsidence of the surface had occurred at the Brier Hill mine, we have no evidence of movement of ground at surface. There could be a small amount of such movement that would escape our notice as the most critical part of the area carried a stock pile for many years, which has recently been cleaned up. There were no accurate established bench marks on the surface in this area and therefore a releveling at this time would serve no purpose.

However, any material subsidence in this level stock-pile ground would be noticeable now. Also, any important subsidence beneath the stock pile while it was there would have caused caving in the stock pile, and no such caving occurred.

Apparently, then, this is a case where subsidence over mining reached close to the surface but caved itself full before breaking ledge and has accordingly not registered at ledge and probably never will register.

The remarks made in the paper and at the meeting as to the possibility of cementation acting as a deterrent to subsidence of loose ground certainly have no application to the Middle Huronian iron formations of Michigan with their enclosing rocks nor to the iron formations of the Upper Huronian, though the pyritic, graphitic, black slates of the Upper Huronian have shown important cementation in short periods of time. Numerous operations in different places on the Cuyuna, Mesabi, Gogebic, and Menominee ranges have penetrated old caved mine workings in iron formation and in ore and in none of these has there been a sign of cementation so far as the writer has been able to observe. The softer and more thoroughly oxidized ores have shown a mechanical packing under pressure, which has made the material almost as resistant to mining operations as the original soft ore would be, but no evidence of cementing of particles has been observed. These workings have stood periods ranging from 15 to 50 years, in different places.

Furthermore, on the Gogebic we have run into the natural stopes and caves caused by the original concentration and reduction of volume in the ore by the removal of the original silica content of the iron formation, leaving the concentrated hematite behind as ore. All of the geological evidence is that this process was virtually complete before the Cambrian sea invaded this area; yet many of these natural stopes and rubbly caves are full of loose, jagged boulders of iron formation entirely uncemented, and forming an extremely unpleasant mass of loose, broken rock to drift through. Also, of course, there is some cementation in these ancient natural caves, though the reverse is more commonly true.

On the Cuyuna and Western Menominee ranges, which are apparently Upper Huronian, cementation has not been noted in caved areas of the formation, though it is noted in the associated pyritic, graphitic slates. The Mesabi, Gogebic, Marquette and Eastern Menominee formations, together with their enclosing rocks, show practically no signs of cementation. My field acquaintance with the Marquette caved areas is not as complete as with the other areas mentioned and therefore there might be a contradiction on this. However, I have not heard of one and would be much surprised if there were one.

Cementation of caved areas is, of course, common and rapid in many of the sulfide areas of the West, but apparently is negligible in the pre-Cambrian iron formations.

W. E. Seppanen,* Vulcan, Mich. (written discussion).—With reference to ground movement, during the summer of 1924, a 12-in. discharge line was installed from the Brier Hill shaft over the area in question.[12] To the knowledge of the writer there never has been any work done in maintaining this line.

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* Mining Engineer, Penn Iron Mining Co., Pickands, Mather & Co.

[12] This pipe line extends along the north side of the stock pile passing over drill holes 1, 2 and 3, directly over the orebody of the seventh level. These drill holes are shown in Fig. 1.—G. S. R.

The area immediately north of the original location of the railroad tracks from 100 to 750 ft. west of Brier Hill shaft was used as a stock-pile ground from the fall of 1928 to the summer of 1933. During the winter of 1928 and 1929 the pile contained 64,234 tons, which was shipped during the summer of 1929. During the winter of 1929 and 1930, the pile stocked 71,657 tons, of which 65,764 tons was shipped in 1930. During the winter of 1930 and 1931 up to March 1, 1931, a pile of 33,527 tons was stocked, which remained on this location until July, 1933. At no time was there any difficulty in maintaining the stock-pile trestle, nor was there any known settling in the pile due to subsidence.

G. S. RICE.—These physical evidences reported by Mr. Seppanen seem to be conclusive that no appreciable subsidence of the surface took place above the critical area of those Brier Hill stopes which underlie the position of the former railroad tracks. Any appreciable local sinkage of a long 12-in. pipe line or of a stock-pile trestle carrying a track would have attracted attention. Mr. Seppanen has sent information in a recent letter that he had surveyed to the top of the casing of 10 boreholes, and making due allowance for the smashing down of the casings to ground level in the nearly 20 years that had elapsed, he concluded from the evidence that had been found that there had been no subsidence of the surface.

Ground Movement and Subsidence at the United Verde Mine

By C. E. Mills,* Jerome, Arizona

(New York Meeting, February, 1934)

Studies of ground movement and subsidence resulting from mining operations cover a broad field. It is also a very important consideration and one that eventually affects nearly every mining operation of any extent. The damage to surface buildings, removal of mine surface plants and replacements of main hoisting shafts must reach an astounding expenditure were these cost figures available over a period of years.

Were the subject one of mathematical formula and dependent only on the angle of draw of the walls or country rocks, the problem would be relatively simple. Consideration must be given to depth and extent of workings; structural conditions of the orebodies and enclosing country rock; physical properties of these same rocks, including schistosity, bedding, jointing, slips, dikes, faults, and other structural weaknesses. The method of mining, extent of openings, supporting pillars, and amount of moisture or water underground are other pertinent factors.

With such a large number of variable factors, it is apparent that the subject of subsidence is one that must be based largely on experience and results at other mines where conditions are somewhat comparable. In presenting this paper, it is the intent to describe the progressive results of past ground movements at the United Verde, to bring out physical conditions largely responsible for this movement, but to refrain from theorizing on the ultimate extent of subsidence resulting from future mining operations.

Geology and Structure[1]

The ore deposits of the United Verde are of the massive sulfide, schist-replacement type. A structural anticline formed by the intrusion of diorite between bedding planes of steeply dipping and severely folded greenstones and porphyry localized the deposit of sulfide. The mineralized zone is of great horizontal extent varying from 200,000 to 400,000 sq. ft., and extends from the surface to the lowest levels at a steep dip of from 50° to 75°. The irregular interfingering of the sulfide with

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* Mine Superintendent, United Verde Copper Co.

[1] For greater detail, see: M. G. Hansen [Geology and Ore Deposits of the United Verde. *Min. Congress Jnl.* (April, 1930)] and T. W. Quayle [Mining Methods and Practices at United Verde. U. S. Bur. Mines *Inf. Circ.* 6440 (1931)].

the schists and porphyry, typical of replacement types of ore deposits, forms an irregular and indefinite footwall boundary.

The sulfide mass varies in horizontal section from elliptical to roughly crescent shaped with a length of about 1100 ft. along the strike and an average width of 300 ft. It lies between the chloritic-schist footwall and the diorite hanging wall. The diorite is fresh (unaltered) and hard with a fairly uniform texture and little evidence of alteration. The black schists consist essentially of chlorite. The quartz porphyry ordinarily is fresh and quite hard although in zones of shearing it may be quite schistose and grade into black schist. Occasionally, near the sulfide body where acid leaching has had its effect, the porphyry is altered, softened and kaolinized.

The main orebodies occur along the contact of the massive sulfide and the schist with smaller commercial ore areas on the schist-porphyry contact or lying entirely within one of the three mentioned rocks. The ore zone at surface is roughly lenticular in shape, 1100 by 600 ft. with a north-south strike. On the lower levels the commercial ore is largely confined to the sulfide-schist contact and averages 1100 ft. along the strike and from 25 to 250 ft. in width. The general cross-section at depth is crescent shaped, with an average strike of N.50° E.

The lenticular shape of the ore zone on the upper levels and the occurrence of commercial stoping areas throughout the sulfide mass was due to the greater extent of the general north-south schistosity and its more definite localization. This, together with the flattened diorite hanging wall above the 1200-ft. level, is responsible for the greater penetration of the ore-bearing solutions into the sulfide mass and the distinctive ore pattern typical of the upper levels.

The crescent-shaped ore zone on the lower levels is due to the preponderant structural influence of the diorite contact which largely limited the commercial ores on the lower levels to the crescent-shaped iron-schist contact.

The massive sulfide is generally very hard and dense and stands well over large areas without timber support. This permits mining by cut and fill or other open stoping methods. The schist varies greatly in strength and hardness over various parts of the mine but ordinarily stopes in these areas require timbering and take weight whenever the ground is wet. The stoping areas within the porphyry footwall are very limited and need not be considered.

Structural Weaknesses

Shearing and Jointing.—Prior to the period of mineralization the pre-Cambrian lava flows and sediments, termed "greenstone complex," were compressed into close folds and tilted steeply to the northwest. This resulted in regional shearing in a north-northwesterly direction. Although faulting occurred both in pre-Cambrian and Tertiary ages—

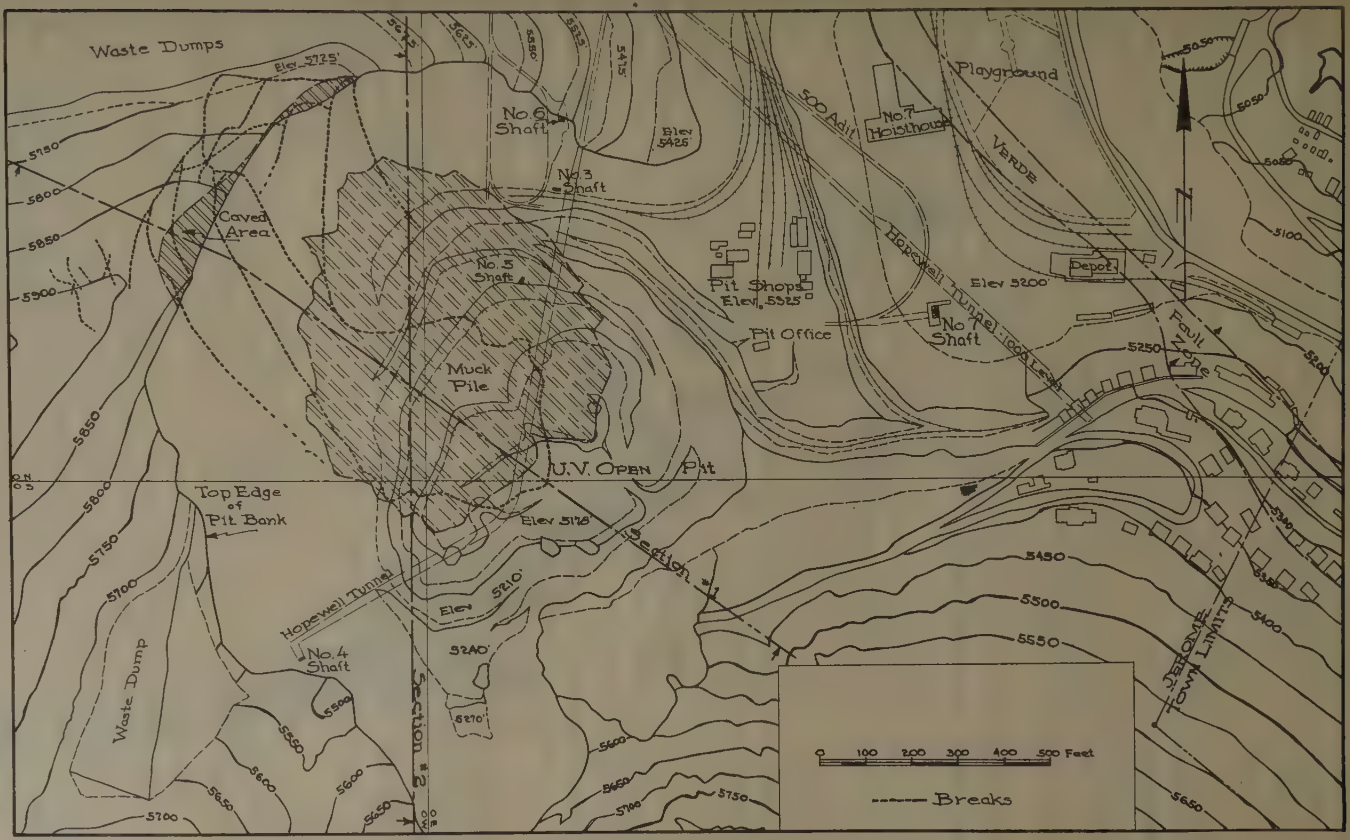

FIG. 1.—SURFACE MAP, UNITED VERDE COPPER COMPANY.
Showing shovel pit, outline of broken muck in pit bottom as of March 23, 1931, fractures in diorite wall and fractures extending back from the upper limit of the pit slope. Surface plants, main shafts, tunnel locations and the Verde fault are indicated.

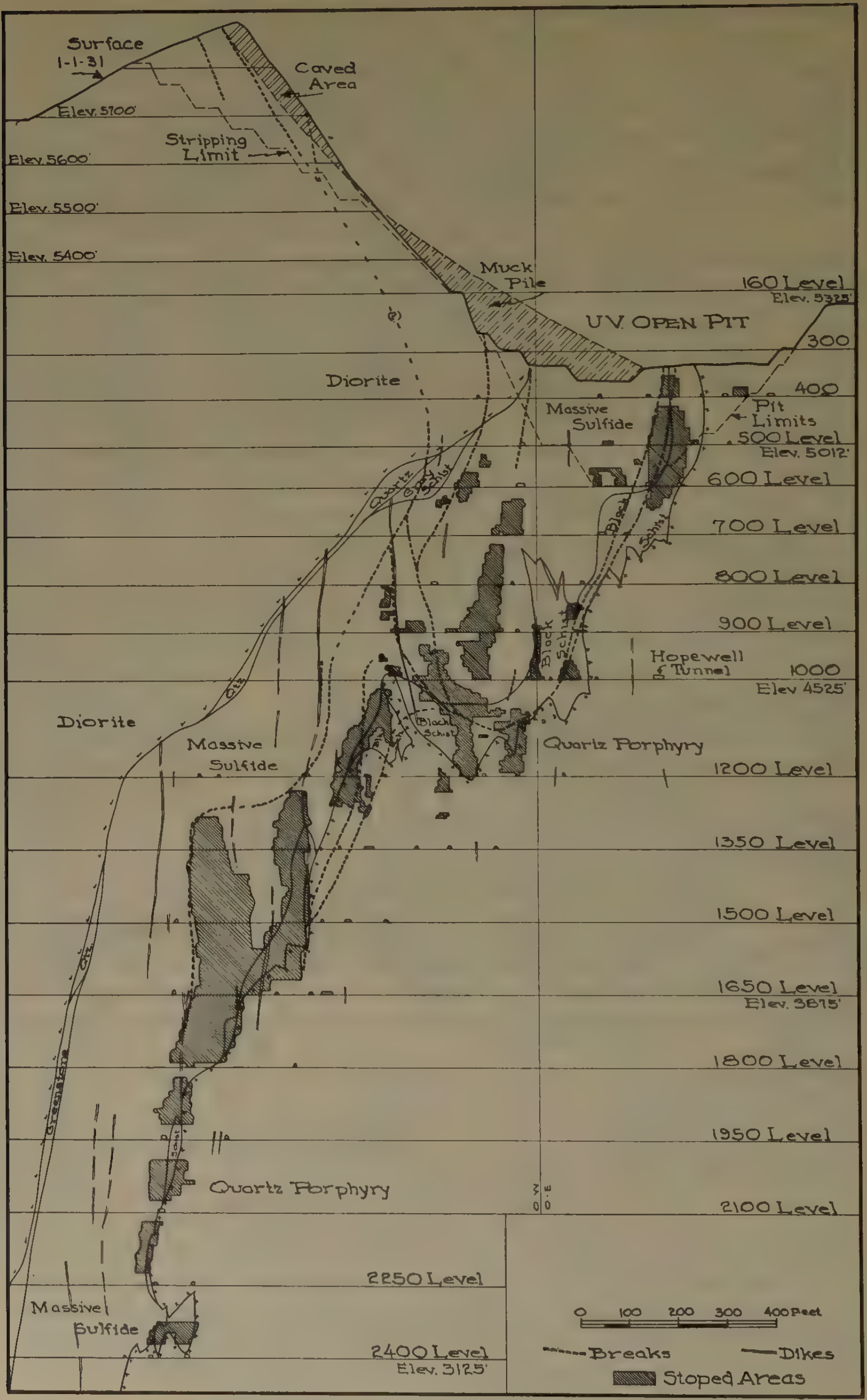

Fig. 2.—Caption on opposite page.

together with some cross faulting—these do not seriously affect the ground movement at the United Verde except that weak planes do exist parallel to the Main or Verde fault, which has a displacement of some 3800 ft. (Fig. 1).

Jointing or slips in the porphyry footwall strike in all directions but most frequently between N.70° W. and S.80° W. The dip of jointing in the porphyry varies from 9° to 84° but dips as low as 35° are prominent, indicating that 35° would be the ultimate draw in the porphyry providing a caving method were used or the openings extensive enough and mining operations extended over a sufficient number of years to reach the maximum subsidence possible.

The distribution of slips or jointing in the diorite is somewhat similar in strike and dip to that of the porphyry. The lower angle dips are not as prevalent in the diorite and dips below 47° are not numerous, so that the ultimate angle of draw in the diorite might be taken at 47° were conditions conducive to and mining extensive enough to give an ultimate or maximum subsidence.

Dikes or "Water Courses."—After the formation of the sulfide mass but before mineralization was entirely completed, a series of small andesite dikes, locally known as "water courses," cut the ore masses and other formations. These dikes have an east-west trend, which is nearly at right angles to the schistosity of the quartz porphyry. They vary in thickness from a few inches to several feet and are often continuous and nearly vertical through several levels, thereby creating definite planes of weakness. These east-west dikes occur extensively throughout the entire mine and where highly altered, as usual in the upper levels, are extremely weak and cause difficulty in stoping operations.

Next in importance to the east-west lines of weakness caused by the dikes are the more or less perpendicular lines of weakness determined by massive sulfide contacts, tongues and stringers of unreplaced country rock in the massive sulfides and also other structures related to the trend of the schistosity. Thus there is a definite tendency for the upper-level sulfide mass to break up into more or less rectangular blocks with nearly vertical sides.

Schist Footwall.—The natural weakness of the schist along the footwall of the massive sulfides, together with the further weakening due to

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FIG. 2.—VERTICAL SECTION THROUGH SULFIDE MASS NORMAL TO STRIKE OF OREBODY. (SEE FIG. 1 FOR EXACT LOCATION.)

Shows sulfide mass with flattened footwall and overhanging diorite hanging wall near 1200-ft. level; also band of black schist between footwall porphyry and iron mass. This schist area on 1200-ft. level is flat in dip; is weakened by stoped areas and offers little support to overhanging masses of iron and diorite. East-west dikes or "water courses" are indicated and form planes of weakness. Breaks are indicated by dotted lines. Iron mass has probably settled away from overhanging diorite between 700 and 1200-ft. levels, but lack of development openings in hanging wall makes this supposition an indeterminant factor.

the concentration of middle level stoping along this footwall, as well as local flattening of the dip, constitutes the most serious and fundamental factor contributing toward weaknesses in relation to ground movement.

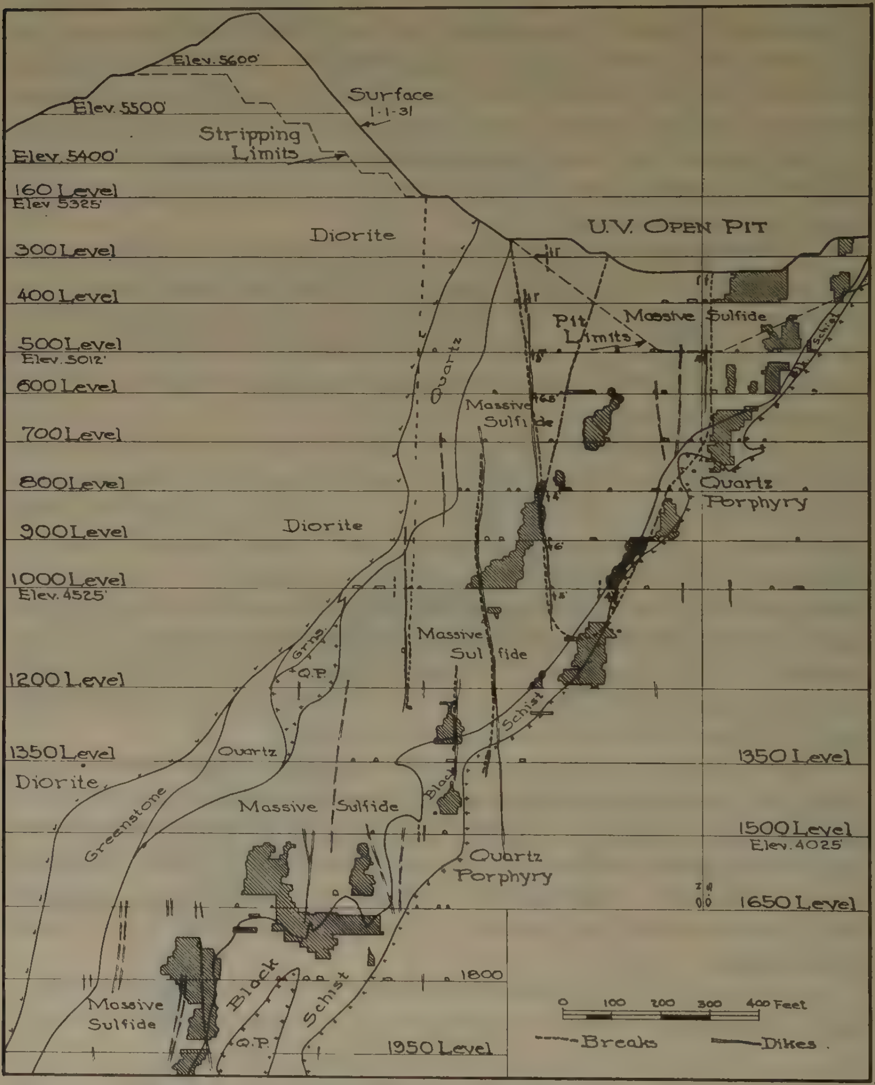

FIG. 3.—VERTICAL NORTH-SOUTH SECTION SHOWING WEDGE-SHAPED BLOCK OF GROUND THAT DROPPED APPROXIMATELY 6 FEET ON JULY 1, 1929. (FOR EXACT LOCATION, SEE FIG. 1.)

Figures adjacent to dotted outline of this block of sulfide represent vertical movement on each level.

Particular attention is called to the flattened dip or offset in the sulfide pipe just above the 1200-ft. level (Figs. 2 and 3). This permits the entire weight of the 700-ft. column of sulfide to bear directly upon the soft schist footwall stoping areas. This schist has not only been weakened by stoping operations but it softens with excessive moisture or water from underground or surface drainage.

It is probable, however, that this weakness of the footwall would not have sufficient in itself to cause serious ground movement were it not for the relatively steep east-west and north-south lines of weakness, which permitted individual blocks of the overlying massive sulfide to crush into the footwall wherever the support was weakened.

PHYSICAL CONDITIONS CONTRIBUTORY TO SUBSIDENCE

The United Verde mine has been developed by 64½ miles of raises, drifts and shafts, all of which are in a limited area laterally. Development data are shown in Table 1.

TABLE 1.—*Development Data, United Verde Mine*

	Drifts	Raises	Shafts	Total
Prior to 1917	101,991	30,261	3,762	136,014
1917 to 1931	131,914	67,812	5,677	205,403
Total footage	233,905	98,073	9,439	341,417

Production to date totals 20,306,736 tons, of which 4,623,790 tons came from the open-pit operations.

Fig. 4 shows diagrammatically the progressive development of the mine in depth by years, the total commercial ore produced by levels and the cumulative production by years.

Mining Methods[2]

The orebody at the United Verde is very irregular in outline both in lateral extent and vertically. No one mining method can be applied to any given level; each stoping area must be carefully planned and a method chosen that is most suited to the physical character of the block of ore in question. The percentage of underground ore mined by various methods during the year 1929 is typical of past operation: Horizontal cut and fill, 63.7 per cent; inclined cut and fill, 5.3; square set, 22.5; inclined square set, 2.3; top slice and fill, 6.8; shrinkage, 1.4.

Above the 1500-ft. level, or where the sulfide mass is more nearly lenticular in cross-section, the commercial ore or stoping areas, although all interconnected and irregular in extent, occur more widely scattered through the sulfide mass. No regular system of vertical pillars was necessary therefore to support the hanging wall, and pillars were limited to the level or sill pillars, which in all cases were mined by square-set methods. Mining prior to 1910 above the 900-ft. level was largely confined to square-set methods.

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[2] For greater detail, see: T. W. Quayle and J. F. Cowley [Development and Mining Methods at United Verde. *Min. Congress Jnl.* (April, 1930)] and T. W. Quayle, reference of footnote 1.

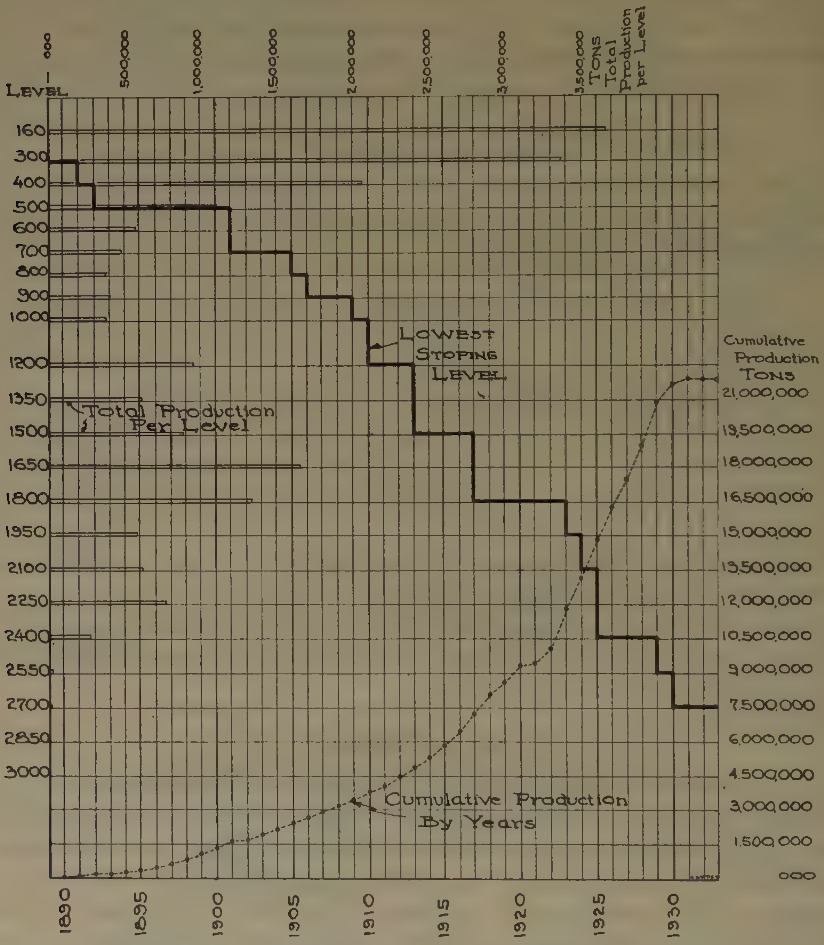

FIG. 4.—PROGRESS IN DEVELOPMENT OF UNITED VERDE MINE IN DEPTH BY YEARS; TOTAL PRODUCTION OF ORE BY LEVELS, AND CUMULATIVE PRODUCTION BY YEARS FROM 1890.

Below the 1650-ft. level the mineralization or commercial ore areas are largely confined to the crescent-shaped iron-schist footwall contact, and the stoping areas, although quite irregular, are continuous for approximately 1100 ft. along the strike of the footwall of the iron mass. In this section of the mine the stopes generally are laid out 60 to 100 ft. parallel to the strike, with 35 to 40-ft. intervening vertical pillars to support the hanging wall. The floor and sill pillars are mined by regular square-set or rill methods and the vertical pillars later sectionalized and removed by square set or top slice and fill.

The removal of vertical and floor pillars in the section of the mine below the 1650-ft. level has not progressed far enough to materially weaken the support of the hanging wall. However, some crushing has taken place in the vertical pillars just below the 1500-ft. level and undoubtedly the continued mining of supporting pillars will result in subsidence or ground movement at this depth.

All mining methods at the United Verde have always included waste fill. Prior to pit operations the waste was mined by glory-hole methods and delivered to each level by a waste-raise system from surface to the lower levels. During recent years pit operations furnished sufficient waste stripping to take care of all underground requirements.

Pit Mining.—The pit program was started in 1917 to uncover the ores above the 600-ft. level, this portion of the mine being within active fire area, which extended from the surface to the 900-ft. level. The major portion of the stripping was in the diorite hanging wall—a hard, dense, coarse-grained diorite and quite strong structurally. The slope was planned on a ½ to 1 or 63° including working benches. (See Fig. 2.) The major stripping program involved approximately 8 million yards of waste and was completed in 1927.

Since then the major ground movement, which fractured the pit wall, has necessitated a change in pit plans, and the slope is now being flattened to a 1 to 1 or 45°, involving an additional stripping yardage of 4 million cubic yards of waste.

Drainage.—The United Verde is a dry mine. The normal volume of drainage above the 1000-ft. haulage level averages 10 gal. per minute and the water below the 1000-ft. level is handled by a 500-gal. Quintuplex pump and averages 180,000 gal. daily.

The average rainfall at Jerome is about 20 in. annually. Occasional severe rainstorms cause considerable surface drainage to pass down the ore-bin raises from the pit to the 1000-ft. haulage tunnel.

The pit proper is protected by surface ditches, which prevent some surface run-off into the pit. However, the area of the pit including the pit wall below the drainage ditches is approximately 64 acres. Although the greater volume of rainfall within the pit proper passes down the ore-bin raises to the 1000-ft. haulage level, an appreciable portion seeps into the stoped areas on the upper levels.

Since the major ground movement has carried through to the surface, the proportion of surface waters reaching the stoped areas underground will increase as the pit deepens. This condition is another factor that possibly may affect future ground movement. Excessive moisture or water has always been regarded as a condition to be avoided in stoping operations, particularly within the schist areas. It has been and is the practice to concrete all ditches directly over the stopes, to prevent drainage water from seeping down into the stopes.

Mine Fires

Prior to 1917 or 1918 ground movements at the United Verde were local in nature and largely due to mine fires. The earliest fire of which there is any record occurred in 1894 on the 300-ft. level in the Hampton stope. The cause was attributed to spontaneous combustion brought about by ground movement. This fire affected an area of about 50,000 sq. ft. on the 300-ft. level and soon extended downward, requiring the sealing off of this section by bulkheads.

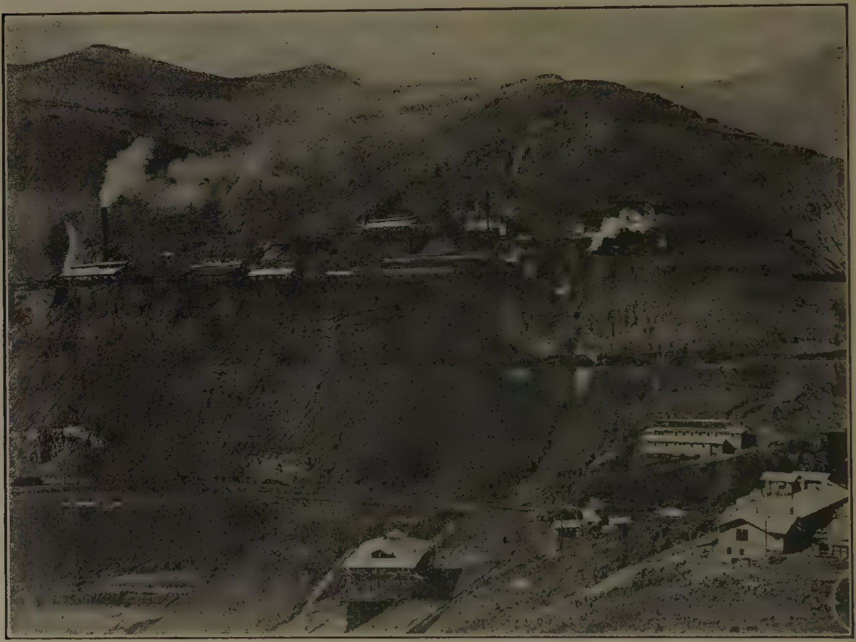

FIG. 5.—GENERAL VIEW OF MINE SURFACE PLANT IN 1917, SHOWING MINE SHOPS, POWER HOUSE, ORE BINS AND SHAFTS 3 AND 4.

Plant was located on 80-ft. level and a portion of it within proposed pit limits. Smeltery formerly was on same level and subsidence, obsolescence and pit operations necessitated construction of new smelter at Clarkdale and mine shops on 500-ft. level.

Sometime later the "Chromes", another group of stopes, with an area of 55,000 sq. ft., took fire on the 300-ft. level, and this fire extended down to the 700-ft. level. These areas were sealed off in 1912.

In 1913 the Middle stopes on the 600-ft. level took fire and soon involved a block of stoping areas from the 400 to the 1000-ft. levels. The average section of this fire area was 30,000 sq. ft. In 1918 the 16-G stope on the 400-ft. level caught fire from red hot sulfide calcines, which dropped through a break in the ground from the level above.

This involved about 10,000 sq. ft. and this stope was immediately sealed off.

The sulfide ores, particularly in the upper portion of the mine, are sufficiently high in sulfur to support combustion, so that as the fire spread the sections involved were sealed by airtight bulkheads to prevent air from reaching the burning sulfide rock. These fire areas were later reopened and stoping operations conducted under air pressure to keep the gases back, but this method of mining was hazardous and expensive. All mining within the fire area was stopped in 1916 and the pit program planned to recover the major portion of the ores was tied up within the fire areas.

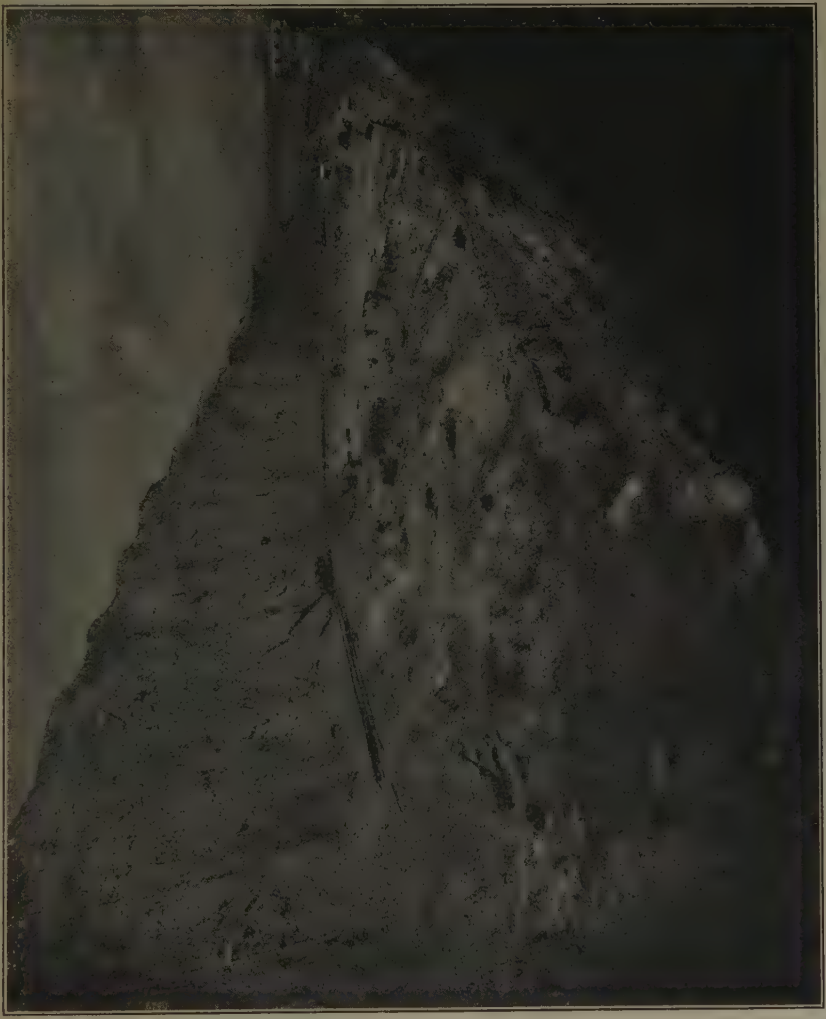

FIG. 6.—GENERAL VIEW OF PIT OPERATIONS IN EARLY PART OF 1927. Eight-yard Marion shovel in center of picture is completing major waste-stripping program above 160-ft. level.

Ground Movements in Past Years

Ground movements during the early years of operation were local in extent and largely caused by fires in the upper portion of the mine. Shaft No. 1 was started in 1883 and extended from the surface to the 500-ft. level. It was within the ore areas and was used until 1894, at which time it was abandoned because of ground movement from adjacent stopes. No. 2 shaft was bottomed on the 700-ft. level in 1900 and later deepened to the 1000-ft. level. It was finally abandoned in 1909, because it was involved in a mine fire on the 250-ft. level.

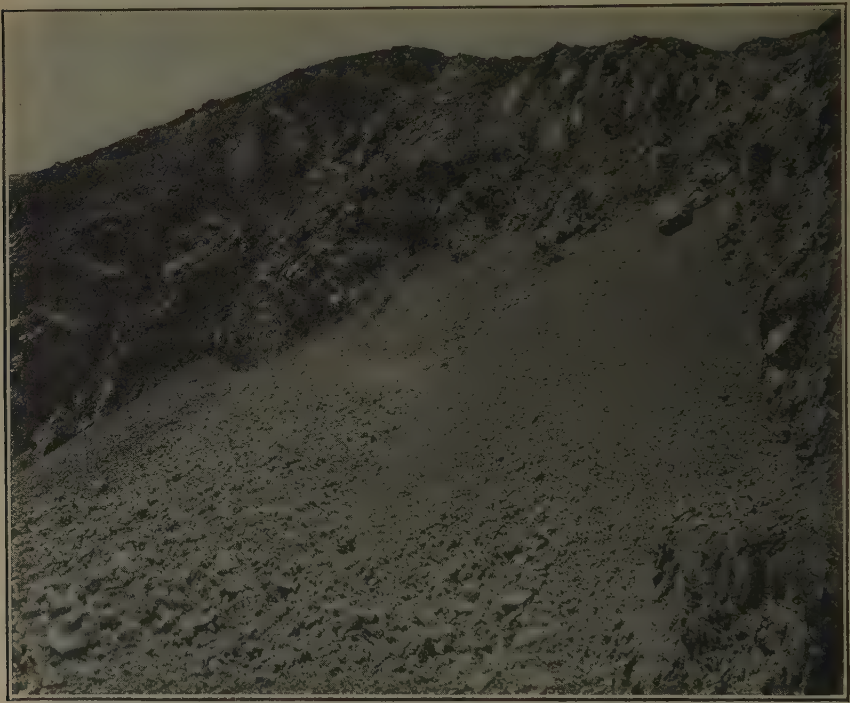

Fig. 7.—Condition of muckpile that slid out of diorite wall of pit on March 23, 1931. Volume of slide approximately 500,000 cubic yards.

Prior to 1915 the mine shops and smelter were on the 80-ft. level directly over the orebody. Settlement due to the workings on the upper level caused considerable trouble in maintaining the power plant and smelter equipment in an operating condition. The shovel program was first contemplated in 1913, so that subsidence and the contemplated pit stripping program was a deciding factor in removing the smelter to Clarkdale in 1915 and the mine surface plant to the 500-ft. level in 1920.

During more recent years a number of minor ground movements occurred in connection with stoping operations, which probably were contributing factors in weakening the supporting structure to the hanging wall. In 1918 a floor pillar above the 1200-ft. Hampton caved, this movement extending upward through the mined areas directly above. In 1922 another floor pillar over the 1350-ft. level 14-M stope suddenly failed, because a large block of iron broke from the hanging wall. Another movement of considerable extent was a settlement of the 15-M stope, which carried through from the 1000 to the 800-ft. level.

In the transition from mining individual stopes on the upper levels to the stope and pillar system necessary on the lower levels, a pillar was lost on the 1650-ft. level, thereby making one stoping area 140 by 250 ft. in cross-section. This stope was mined up to the eleventh floor when caving commenced. Top slicing with fill was resorted to in order to recover the remaining ore in this block and speed up the rate of mining. The top of this lense extended to the 1200-ft. level, where the waste fill was introduced. However, the drag of this waste fill on the walls and the mining of the large 10-O orebody on the 1650-ft. level weakened the support to the hanging. (Fig. 2 shows lines or breaks above this 10-O orebody.)

MAJOR GROUND MOVEMENT AND SUBSIDENCE

No. 5 hoistroom is on the 1000-ft. level. It was constructed in 1916 before full geological information was available as to the structure and extent of the United Verde orebody, and it was located directly on the contact of the schist and the sulfide.

A small amount of movement in the floor of this hoistroom was apparent in 1925. This movement became more pronounced and finally cracks and displacement occurred in the concrete lining of the hoistroom. Constant attention was required to keep the shafts of the hoist and the generator set in alignment. Early in 1929 the hoistroom movement became so serious that the back over the hoist had to be supported by steel girders and steel sheds placed over the electrical equipment. Finally timber bulkheads were erected to support the concrete lining over other parts of the hoistroom. A new hoistroom in the footwall was started in April, 1929, and the hoist moved to the new location in August, 1930.

During 1925 and 1926 mining operations in the lower pit were extremely hazardous and severe on workmen because of the gas from the fire areas. The SO_2 gas was not only strong but contained small percentages of H_2S. Calcined sulfides in hot areas would burst into flames when picked up by the shovels. A dust explosion in No. 7 ore-bin raise in May, 1926, blew out doors on the upper levels and resulted in three fatalities.

Early in 1928 an effort was made to cool the fire area by the introduction of slimes over the hot sections in the pit and also through bulkheads and diamond-drill holes underground. All precautions were taken to

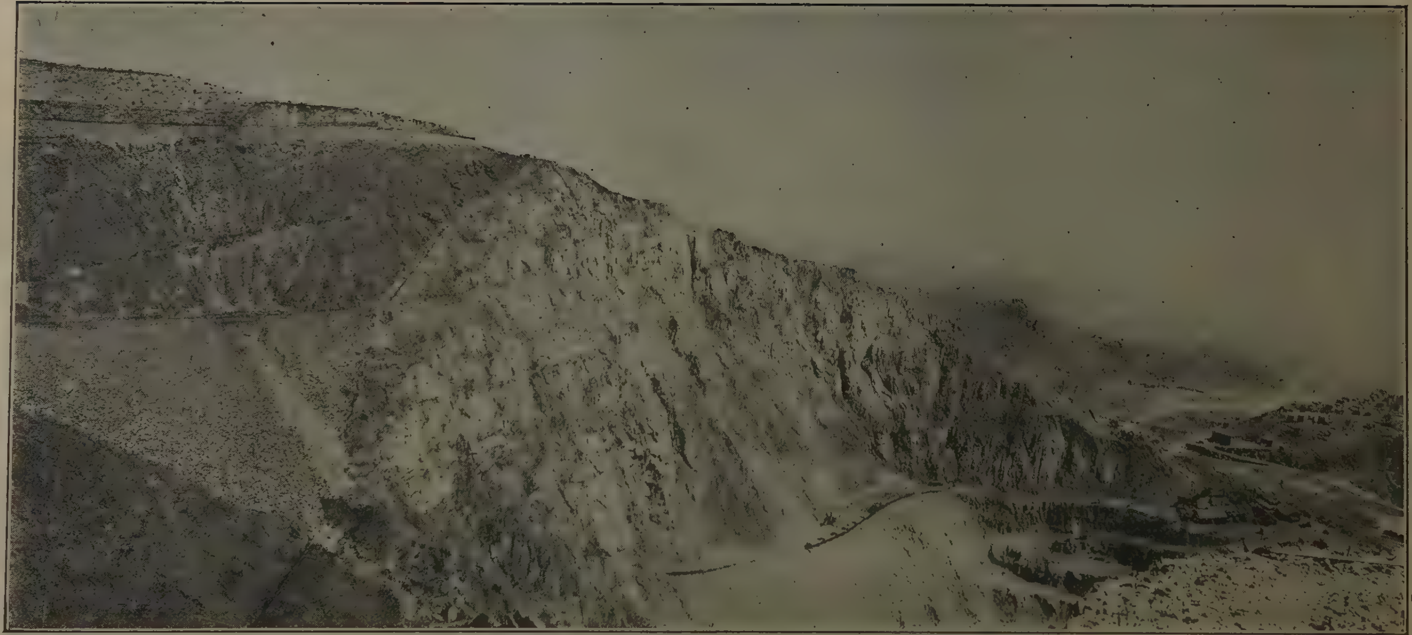

FIG. 8.—GENERAL VIEW OF DIORITE WALL ON NORTH AND WEST SIDE OF PIT, TAKEN JULY 1, 1931, OR SHORTLY AFTER WORK OF CLEANING UP SLIDE HAD STARTED.

Shovel in lower foreground is a 120-B removing slide material on 160-ft. level. Bench being cut above upper pit wall is at an elevation of 5935 ft., or 610 ft. above 160-ft. level. Upper bench is upper limit of present additional stripping program that will flatten slope of pit walls to one to one.

keep the amount of water as low as possible and also to drain excess water into the regular mine-drainage system. Approximately 16,000 tons of solids were introduced prior to interruption by a serious mine fire.

On Jan. 13, 1929, a fire broke out in the 16-J-K timbered stopes on the 1000-ft. level and soon extended to the 800-ft. level, involving a horizontal stoping area of 10,000 sq. ft. The fire was fought by means of water and later by the introduction of slimes through available openings and diamond-drill holes. This fire was not entirely out until March 12 of the same year.

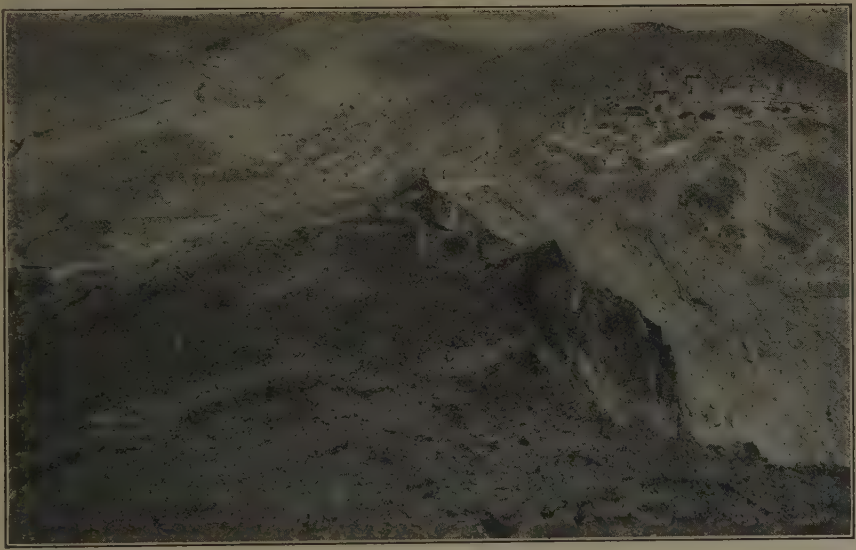

Fig. 9.—View taken at rim of pit showing subsidence above pit shortly after major ground movement in July, 1929.

During this fire a large volume of water and slimes was introduced and as this area was in the soft schist footwall the excess water was undoubtedly the direct cause of the movement that soon followed.

Shortly after this fire an appreciable settlement was noticed on the east-west "water course" about 200 ft. south of the old No. 5 hoistroom, and on July 1 a movement of 4 to 6 ft. occurred within the space of a few hours. This block of ground was wedge-shaped in vertical section, the north side following the almost vertical "water course" and the south side at a dip of about 65°. The north-south extent on the 1000-ft. level was about 120 ft. and this break extended through to the pit where the wedge section gave an added north-south dimension of 350 ft. (See Fig. 3.)

The movement was taken up on the east or footwall side by the schist stoping areas and the west boundary or hanging-wall side is not definitely established, because of lack of development openings in the hanging-wall diorite. Although some of the breaks may be observed cutting through the outlying tongues and irregularities in the diorite hanging wall, there is a manifest tendency of the deeper breaks to follow the general trend of the hanging wall, when they reach the diorite, rather than to continue vertically upward into the main mass of overhanging diorite.

The movement on the north side, where limited to the east-west "water course," shows an abrupt displacement of from 4 to 6 ft. Although there is some lack of uniformity in the movement, in general the block settled as one mass. Sufficient warning was given during the two days preceding this movement so that the stoping areas within or adjacent to the ground involved could be abandoned. The outline of this break is shown on Fig. 1.

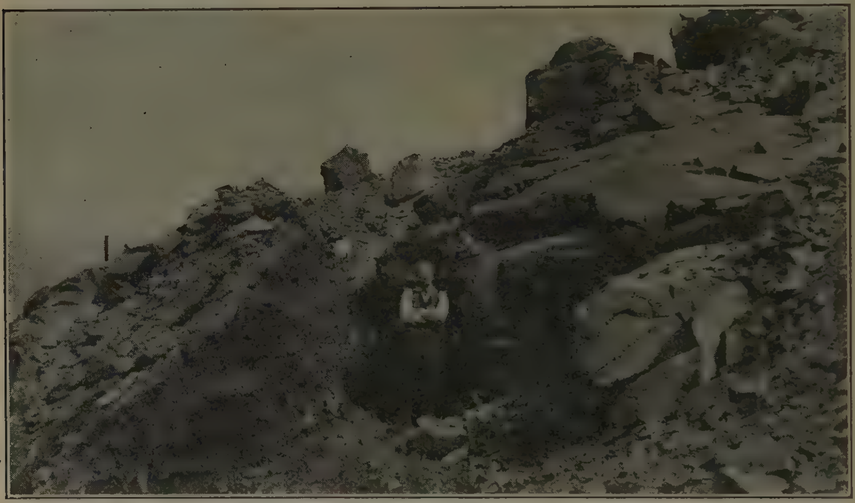

Fig. 10.—View taken on north side and above pit, showing tilting effect of block movement.

Block where workman is standing had dropped down or block adjacent to rim of pit had tilted, resulting in apparent downward displacement of block farther away from pit wall.

Slide in Pit

The ground movement underground on July 1, 1929, did not seriously affect pit operations for some time. Although the pit walls were fractured and considerable settlement occurred above the pit, the diorite wall on the west side still retained considerable strength.

The breaks above the pit showed a displacement of from one inch to as much as 12 ft. adjacent to the top of the pit wall. (See Fig. 9.) Although the breaks generally showed a downward displacement on the side toward the pit, in one or two instances the movement was just the reverse, indicating that certain blocks had dropped more than others or that some tilting or horizontal movement had occurred in the upper portion of the diorite in the pit walls (Fig. 10).

Early in 1931 the diorite wall of the pit began to work along the breaks that occurred in July, 1929. On March 23, 1931, this section of the pit wall suddenly gave way and approximately 500,000 cu. yd. of broken diorite slid into the lower pit (Fig. 7).

Had conditions in the copper industry continued favorable, this slide would have interfered seriously with pit mining. A lower production rate in the latter part of 1930 and entire cessation of production in May, 1931, offered a favorable opportunity to strip the broken pit wall back to a one to one slope and clean the pit of broken diorite. This program is still in progress and will be completed during 1934.

Effects of Ground Movement

Mine Plant.—The entire smeltery was moved to Clarkdale in 1915 and the mine shops to the 500-ft. level in 1920, so that recent or further subsidence due to mining operations at the United Verde will not cause added expense for maintenance of surface plants. This is not to be confused with the general movement in the Town of Jerome, which has affected some dwellings and physical property owned by the United Verde Copper Co. This particular ground movement is in no way related to the topic under discussion.

Mine Shafts.—As stated before, the early shafts, No. 1 and No. 2, were abandoned because of fires and local ground movement in adjoining stoping areas. No. 3 shaft was started in 1900 and extended to the 1950-ft. level. No. 4 shaft was started in 1910 and extended to the 1000-ft. level. Both of these shafts were timbered and located within the limits of the proposed pit. They were abandoned in 1921. No. 5 shaft, the main ore-hoisting shaft extending from the 800-ft. level to the 3550-ft., was started in 1915. No. 6, the service shaft, was started in 1919 and extends to the 3000-ft. level.

Although the present operating shafts are not seriously affected by subsidence, their location is somewhat insecure and continued underground mining over a period of years will undoubtedly involve these two shafts also. For this reason a new combined service and hoisting shaft was started in 1929 and will be completed before shafts 5 and 6 are seriously affected by continued ground movement.

Mining Operations.—Ground movement has increased the cost of underground mining during the past years to a limited extent and future movement undoubtedly will involve considerable ore reserves on the upper levels, possibly to an extent where their extraction proves unprofitable.

Because of the fire zone in the upper part of the mine, it has been impossible to clean up the ores on the upper levels, so that practically all levels from the surface down contain ore reserves that must be mined by underground methods. This also applies to border ore outside of the pit limits that cannot be mined until the pit ores are exhausted.

CONCLUSION

Although a study of the structural conditions of the United Verde sulfide mass may show definitely the reasons for subsidence, it is much more difficult to predict the extent of further settlement in the next 10 or 20 years.

In flat-lying coal beds and relatively shallow porphyry deposits, the areas involved by subsidence may be fairly well predicted by a study of the joints, slips and structural weaknesses of the orebody and surrounding country rocks.

Such predictions are almost impossible with a deep-seated or vein type of orebody that is several thousand feet deep. Should the conventional method of determining the angle of joints or planes of weakness be used and applied to the 3000-ft. level or lowest workings, the surface areas involved would be so extensive that shafts or surface plants could not be located on the property and development drifts and haulageways to the orebody would be impractical.

The time element is also of great importance. During the coming century or longer, it may be possible that the low-grade sulfide will be mined for its iron, zinc or other byproducts. Over this period of time subsidence may reach or approximate its ultimate possible extent.

During the life of this mine, based on its commercial copper ores, the time element should not be sufficient to permit maximum settlement to lines of ultimate draw. A close check on fractures underground shows that readjustments have taken place since the major displacements in July, 1929, and fractures are becoming more prominent at greater distances from the breaks.

It is also possible and entirely likely that future underground mining operations will involve greater blocks of the diorite hanging wall of the pit. For this reason it is planned to expedite pit operations upon resumption of copper production. If the pit ores can be exhausted within the next six or eight years, mining may not be interrupted by further movement of the pit walls. After completion of the pit, further settlement of the diorite hanging wall will have no serious results, as the surface ground is of little or no value.

Subsidence of the porphyry footwall would involve the new No. 7 shaft if the possible lines of weakness were extended down to the lower levels. The new shaft is in the footwall porphyry and just as far away from the orebody as the property lines and the Verde fault will permit. It is thought that the maximum angle of draw in the footwall porphyry will not extend below the schist stoping areas on the 1200-ft. level during the active life of the mine.

ACKNOWLEDGMENT

Acknowledgment is made to L. E. Reber, Chief Geologist of the United Verde Copper Co., for his helpful criticism and suggestions in regard to the portions of this paper that deal with geology and structure, and to Mr. R. E. Tally, President of the company, for general suggestions on the entire paper.

DISCUSSION

(George S. Rice presiding)

L. E. YOUNG,[*] Pittsburgh, Pa.—What is the source of material for the stope filling?

R. E. TALLY,[†] New York, N. Y.—From waste raises through to the surface. Of course, since the shovel work was started in 1919, the bulk of the filling came from the stripping operations.

H. G. MOULTON,[‡] New York, N. Y.—Where was the old smelter site?

R. E. TALLY.—The old smelter site was directly over the orebody, at an elevation of 5430 ft. and 80 ft. above the top of the orebody. It was built on a slag fill. The ground settlement was excessive owing to underground stoping and mine fires, and the settlement reached a stage where the plant and equipment could not be safely and economically maintained. It was not an uncommon occurrence to see a converter blowing engine setting at an angle of 45° from horizontal. An exploration campaign started in 1908 eventually resulted in the development of sufficient ore reserves to warrant the construction of a new and modern smeltery. This new smelter was blown in in 1915 and the old one closed during that year.

J. A. CARPENTER,[§] Reno, Nev.—When did that sudden cave of large block occur?

R. E. TALLY.—That was in 1929, about two months after the fire on the 900 level, in the schist area previously referred to.

J. A. CARPENTER.—Was there water underground?

R. E. TALLY.—The mine is relatively dry but excess water was used in extinguishing the fire. Water in schist makes such ground very heavy, and it was the water that caused the trouble.

L. E. YOUNG.—At such low levels is there difficulty in holding the stopes?

R. E. TALLY.—No, none whatever. As I mentioned, about two-thirds of the stopes are open mining; occasional pillars 40 ft. wide across the large stopes are used. The schist stopes, however, are heavier, usually requiring timber, and water in these stopes results disastrously.

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[*] Vice President, Pittsburgh Coal Co.

[†] Vice President, United Verde Copper Co.

[‡] Consulting Mining Engineer.

[§] Professor of Mining, Mackay School of Mines, University of Nevada.

G. S. Rice,* Washington, D. C.—In the lower part of the mine, are there natural barren areas to give support to the very heavy ground that rises above the hanging-wall side?

R. E. Tally.—As we go deeper the orebody flattens to a crescent shape, and this change will automatically take care of the weight problem. Also, horizontal pillars are left at frequent intervals for this purpose.

In 1894, at this location on the 300 level, the first fire started. The stope was very large. The formation is massive sulfide of the schist placement type. In some cases the replacement is complete; otherwise it is partial, in which the copper is in the form of chalcopyrite in a schist gangue. Such ground frequently requires timbering. Water makes it dangerous for open-stope mining.

D. Harrington,† Washington, D. C.—What was the cause of the last fire, the one that preceded this ground movement?

R. E. Tally.—I do not think anybody knows. There was no reason in the world for a fire there from spontaneous combustion, and it was not caused by blasting.

L. E. Young.—At a depth of 2000 ft., what would be the approximate diameter in the plan across the orebody?

R. E. Tally.—I could not give the dimensions offhand at the 2000 level, but at the 1650 it has an area of about 400,000 square feet.

L. E. Young.—What does that represent, roughly, in diameter?

R. E. Tally.—About 800 by 500 ft. One stope on this level was about 600 ft. long by 200 ft. wide and had three 40-ft. pillars across the stope.

L. E. Young.—Is there any possibility of the measures supporting themselves at a depth of 1500 feet?

R. E. Tally.—In stope mining we take the largest section that can be mined safely. The walls and roofs of the stopes are usually good. The particularly high-grade ore is heavier than the medium or lower grade material. Schist stopes are considerably more treacherous than the sulfides.

I wish to emphasize that with such large stopes we have not only the regular angle of draw to contend with, but also numerous dikes and other planes of weakness. Most mines have similar conditions in one form or another.

Developments are being carefully checked, because now that we have such a serious ground movement we hope to capitalize it in a mining way.

There are horizontal pillars left at certain elevations, that can be mined at a later date. The ores are continuous with depth and the large vertical and lateral dimensions mined and filled are in themselves serious lines of weakness. A ground movement would have occurred in the course of time from these operations, aside from the fire and water referred to. These conditions, however, accentuated the cause and hastened the result.

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* Chief Mining Engineer, U. S. Bureau of Mines.

† Chief Engineer, Safety Division, U. S. Bureau of Mines.

Subsidence Following Extraction of Ore from Limestone Replacement Deposits, Warren Mining District, Bisbee, Arizona

By Carl Trischka,* Bisbee, Arizona

(New York Meeting, February, 1934)

During fifty-three years of mining operations in the Warren mining district, the mineralization has been found to cover an area roughly 2 miles long by 1½ miles wide. Ore extraction from the richer portions of this area has been closely followed by surface subsidence.

At the western end of the district, where the ore outcrops at the surface and oxidation is more complete over a wide area, intense surface disturbance is visible in the form of closely spaced open cracks and escarpments. As work has progressed toward the east, more unoxidized orebodies have been found and mining has been at increasing depth, for which reasons surface subsidence is less intense and subsidence blocks become larger and in general are bounded by open cracks, which persist over long distances. There may also be traced on the surface what appears to be a boundary crack, running in a northwest-southeast direction, marking the present southerly limit of subsidence of the district as a whole. It is not completely developed, and has not as yet appeared over the very deep mining operations in the extreme easterly portion of the district. Check surveys show that vertical and horizontal dislocation is still taking place.

All of the various mining operations, including open-pit work in porphyry ore and underground mining of limestone replacement and porphyry ores, have had their influence in disturbing the surface. As these influences frequently overlap, and because the surface disturbance is continuous over practically all ore occurrences, it is virtually impossible to choose an area sufficiently isolated to ascribe the amount of subsidence to any definite volume of ore removed.

After due consideration of all factors, there has been selected for discussion an area 1400 ft. long by 1200 ft. wide, which will be referred to as the Queen Hill block. It is bounded on all four sides by faults, which tend to limit subsidence disturbance to that caused by ore mined from within the block, and likewise to prevent subsidence cracks due to mining outside of the block from entering the area.

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* Chief Geologist, Copper Queen Branch, Mines Division, Phelps Dodge Corporation.

The map (Fig. 1) shows areal geology, including boundary faults, major subsidence cracks, and the outlines of ore mined as projected on a horizontal plane.

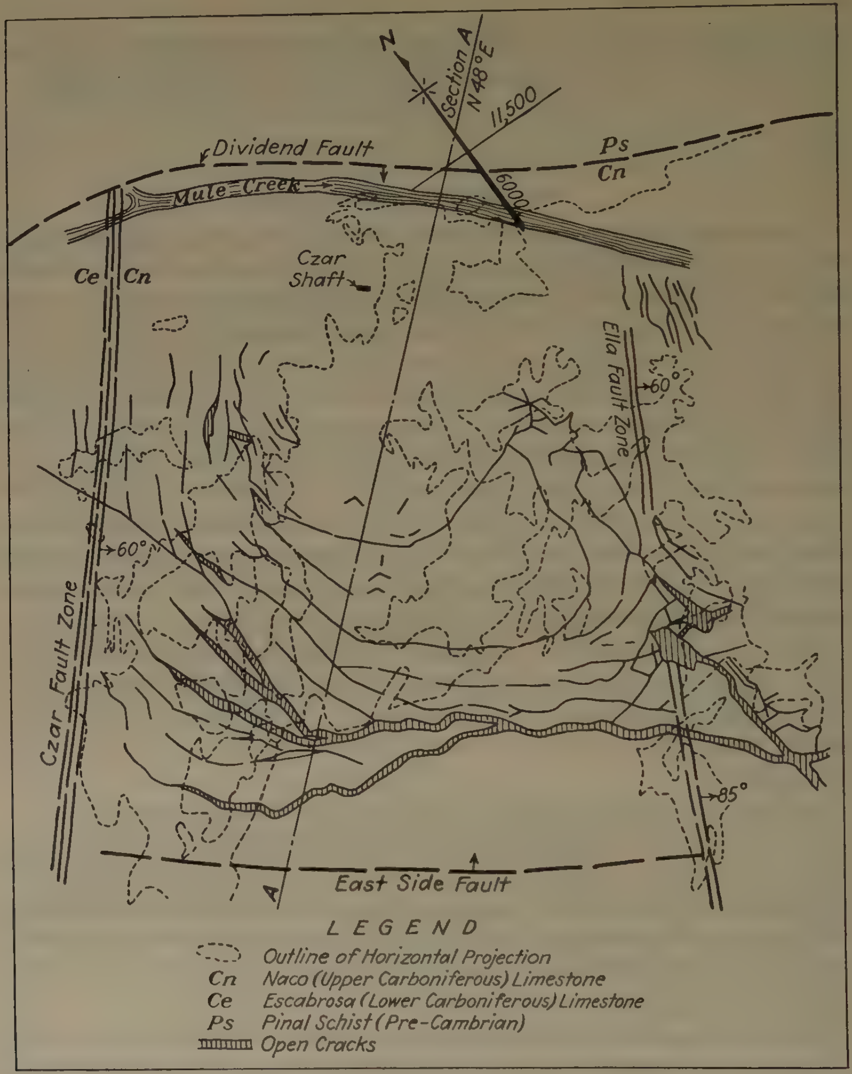
N
Section A
N48°E
11,500
6000
Dividend Fault
Ps
Cn
Mule Creek
Czar
Shaft
Ce
Cn
Ella Fault Zone
60°
60°
Czar Fault Zone
85°
A
A
East Side Fault
LEGEND
Outline of Horizontal Projection
Cn Naco (Upper Carboniferous) Limestone
Ce Escabrosa (Lower Carboniferous) Limestone
Ps Pinal Schist (Pre-Cambrian)
Open Cracks

FIG. 1.—MAP OF QUEEN HILL BLOCK SHOWING CRACKS, BREAK ZONES AND HORIZONTAL PROJECTION OF OREBODIES.

The volume of subsidence throughout the Queen Hill block is not obtainable, owing to the lack of topographic maps prior to mining operations. Comparison of the present surface, as shown on Fig. 2, with that taken from maps made in 1908 shows a general subsidence throughout most of the length of the block. The scale of the original maps is too small, however, to make of any value a comparison of volume of subsidence with volume of ore excavated.

Ground settlement was noted early in the history of the district. Dr. James Douglas in 1912 remarked[1] that "settlement of ground over mining operations has been going on ever since mining operations began."

GEOLOGY

The geology of the district is so well known that only references to it relating directly to the Queen Hill block will be made here. The surface of the block (see Fig. 1) is Naco limestone (Upper Carboniferous). Adjacent to the block, on the west side of the Czar fault, is Escabrosa limestone (lower Carboniferous). Pinal schist (pre-Cambrian) is found on the northeast side of the Dividend fault. The displacement on the Dividend fault in this area has been estimated at 3000 feet.

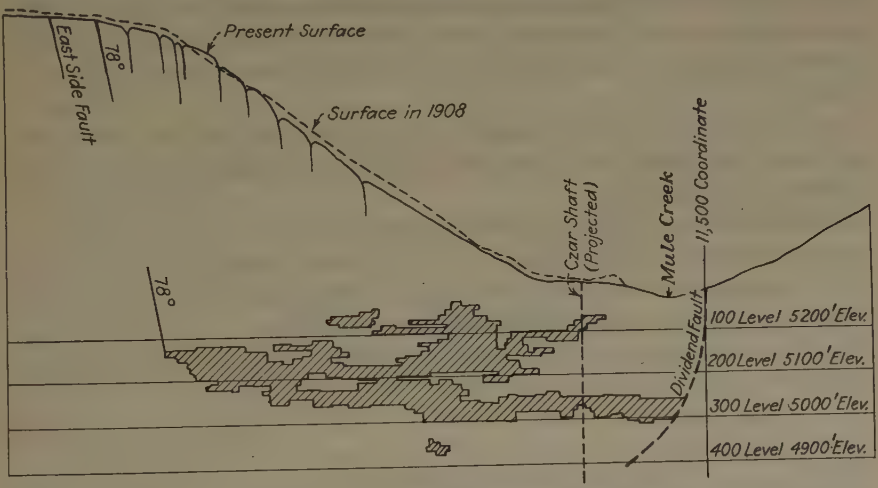

FIG. 2.—SECTION *A-A* OF FIG. 1. LOOKING NORTHWEST THROUGH OREBODIES PROJECTED 100 FT. EACH SIDE OF SECTION.

The Czar and Ella faults belong to the northeast-southwest system of faults, which are believed to be the channels along which ore solutions traveled from the Dividend fault to replace the limestones and form ore deposits. The orebodies in the block are found in the upper Abrigo (Cambrian), Martin (Devonian) and Escabrosa limestones. They are numerous and generally tabular in shape. Associated with the orebodies throughout the block there are large areas of intensely oxidized ground over which pre-mining subsidence, or "slumping," occurred, which influenced later disturbances due to mining. Despite widespread underground oxidation, surface gossans are few.

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[1] J. Douglas: The Copper Queen Mines and Works, Arizona, U. S. A. *Trans.* Inst. Min. and Met. (1912–1913) **22**, 532.

The ore mined from this area came from depths of from 70 to 700 ft. below the surface—the depth being governed principally by topography. The total was about one million tons. The average thickness of ore mined throughout the area of the horizontal projection is 21 ft., which represents 1.63 tons per square foot. In the vertical section (Fig. 2), mine workings have been projected from 100 ft. on each side of the section line.

PRE-MINING SURFACE DISTURBANCES

The limestones of the Queen Hill block were subjected to many distorting influences before mining operations started, caused by the following forces, which have all had their part in guiding and producing most of the present prominent surface cracks opened by subsidence due to the removal of ore:

1. Bedding planes and breaks or slips normal to them are numerous, and are the result of conditions during limestone deposition and regional disturbances.
2. The faults of the northeast-southwest system are the results of regional disturbances before and after the introduction of the ore.
3. Breaks and faults parallel to the Dividend fault are common and are due to movements and adjustments along the fault.
4. Troughing of the limestone bedding created folds and cracks during movements on the Dividend fault.
5. Post-Cretaceous tilting has dipped and cracked the beds, and reopened old faults.
6. Oxidation slumping,[2] of which there is abundant indication, has formed cracks which later were cemented by calcite.

Oxidation slumping, which no doubt was responsible in a large measure for the rapidity with which subsidence followed mining operations, is, briefly, the result of the removal of iron and sulfur during the oxidation of the sulfide orebodies. The orebody shrinks in size, causing a cave to form above it, from which breaks extend upward and (in shallow deposits) finally reach the surface. Further subsidence is caused by oxidation and crushing of siderite, which is formed below the ore during the first stage. Surface cracks thus formed have been cemented by aragonite. In the Queen Hill block, there are so many oxidized orebodies over which oxidation slumping has occurred that the cracks caused by individual orebodies are not separable.

GENERAL

During the period from 1880 to 1913, mining was done largely by the square-set method. Top slicing then replaced the square-set method in the Queen Hill block. During times of high metal prices, the old square-set areas have been mined a second time, using the top-slice method. There have also been intervening periods of several years during which operations have ceased entirely in the area.

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[2] E. W. Wisser: Oxidation Slumping. *Econ. Geol.* (1927) **22.**

In the early square-set stopes, very heavy timber was used and close filling of mined-out sections was carefully followed out. Despite these precautions, supports frequently were inadequate and caving of the overburden occurred. There is little doubt that in many cases this was due to the presence of pre-mining lines of weakness extending upward for considerable distances above the stopes. As the square-set stopes spread over larger areas, subsidence extended to the surface. Close filling reduced the subsidence but could not prevent it.

With the adoption of the top-slice method, the tendency toward rapid subsidence of the overburden after the ore was extracted contributed much to the success of the system. After reaching the surface, subsidence proceeded slowly coincident with mining and without the sudden development of deep holes.

Immediately above the stopes a jumble of rock fragments is formed, which becomes more orderly toward the surface. Over the comparatively shallow deposits of the Queen Hill area, the surface resembles a jigsaw puzzle pattern, whereas over the deeper deposits, subsidence cracks bound larger blocks.

Faults and Cracks in the Queen Hill Block

The southwesterly boundary crack (Fig. 3), the most prominent in the block, is an open crack, 10 to 12 ft. wide in places, and shows a surface subsidence of 3 to 8 ft. It is almost parallel to the Dividend fault, and probably is a part of the East Side Fault zone. This crack is thought to have been caused primarily by oxidation slumping and to have been reopened by subsidence due to mining. Calcite cementing material is found along one wall or the other for several hundred feet.

The Czar fault zone not only stops most of the cracks on the northwest side of the block but also guides them. One exception is the large crack which ends in the Queen glory hole, where one of the earliest ore discoveries was made. The Czar fault dips at 65° to the southeast, and shows some displacement as well as openings along the cracks.

The Dividend fault is difficult to trace on the surface because it is covered by detrital material. It is a wide zone, and hanging-wall disturbances have been noted. The fault dips to the southwest. The area between it and the Czar shaft does not show cracks because it is covered with surface wash in which the cracks soon heal up.

The schist, which forms the footwall side of the fault, has been broken to only a very slight degree. A few irregular cracks, without system or pattern, fade out to the northeast a short distance from the

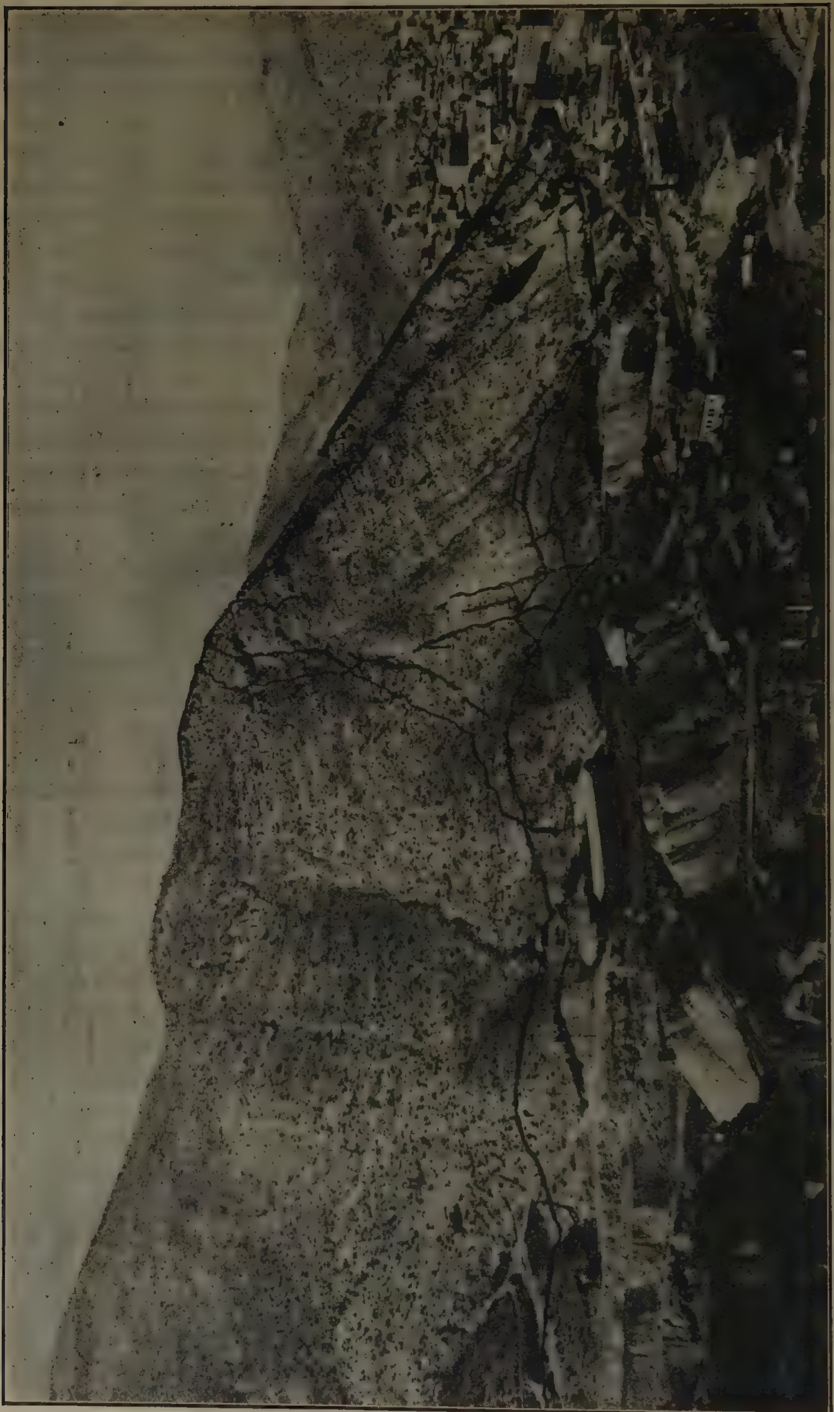

FIG. 3.—MAIN FRACTURE LINES WITH SUBSIDIARY CRACKS, QUEEN HILL, VIEWED FROM THE EAST.

fault and show no displacement. Dwellings on the schist outside of the actual fault zone show no movement despite proximity to mining. From this and the fact that fill has been dumped into Mule Gulch from time to time, because of subsidence, it is evident that the Queen Hill block has dropped along the Dividend fault because of mining operations, and that it did so without appreciable damage to the footwall side.

Outside of the boundaries given above only small cracks with slight or no displacements are found, and are, as commonly found elsewhere, caused by small ground adjustments following displacements on major boundary zones.

Although the Ella fault does not entirely limit the zone of breaks on the southerly side of the Queen Hill block, its influence is very marked. Its failure to limit the subsidence is due to the mining of orebodies that are continuous with those of the block under discussion. The extent of this outer subsidence is obscured by surface wash and dump material.

It should be noted that the faults mentioned are not confined to definite lines but are in reality zones 20 ft. or more wide.

The cracks form a partial halo around the center of the block, owing to the influence of mining on the various intersecting and bounding break zones.

During subsidence, there are formed what appear to be curved cracks. These cracks are not actually curved, but form at the corners of intersecting break zones, and are a series of almost straight cracks which zigzag from one break zone to the other on slips. They start as long cracks of one zone, and follow by tranverse slips around the turn, gradually becoming longer cracks of the next zone, to which they finally become parallel. Two such apparently curved corners are shown on the map (Fig. 1); one at the intersection of the Czar fault and the branch of the southwestern boundary crack, and the other at the intersection of the Ella fault with the same boundary crack. Similar conditions probably prevail at the other corners, which are obscured by surface wash and dumps.

The central slumped area, with its more or less concentric cracks extending outward to definite encircling and bounding cracks, is the common pattern found over mined-out orebodies.

On the section (Fig. 2) only the principal cracks are shown. Minor cracks are very numerous, especially up the slope from the Czar shaft for a distance of 700 ft. The section also shows the original surface as taken from the maps of 1908.

SUMMARY

1. Boundary fault zones limit quite definitely the surface subsidence of the Queen Hill block, hence such zones might form a basis for predicting the extent of surface subsidence.

2. Pre-mining lines of weakness tend very strongly to guide subsidence cracks after mining begins.

3. Owing to oxidation slumping, the overlying limestones were under strain at the time mining was begun, hence subsidence followed very closely upon mining.

4. Angles of break on the boundary lines probably coincide with the dip of the boundary faults. Angles of break on interior cracks vary widely, because they represent overlapping influences of different underground workings.

5. What appear to be curved cracks are in reality the result of intersecting break or fault zones.

6. Subsidence following mining operations must be taken into account, not only as to its effect on the surface where roads, buildings and other equipment must be preserved but also in preserving shafts by establishing circles of varying diameters around them, within which no ore is extracted until the shaft is ready to be abandoned.

[For discussion of this paper, see page 191.]

Surface Subsidence over the Porphyry Caving Blocks, Phelps Dodge Corporation, Copper Queen Branch

By W. H. Kantner,* Bisbee, Arizona

(New York Meeting, February, 1934)

In this paper, no attempt will be made to theorize on subsidence. Only known data and actual facts will be given, with a few exceptions noted where other factors and outside influences tend to change subsidence directly resulting from porphyry caving.

The area under observation is known as the Southeast Extension of the East orebody. The orebody is irregular in outline and dips to the south and the east, extending in depth from about 425 ft. below the surface to 1050 ft. below the surface. In the upper horizon, the orebody is about 400 ft. long and 150 ft. wide. At 720 ft. below the surface, the ore outline is approximately 450 by 170 ft.; 70 ft. below this, it is 500 by 110 ft., with an extension at the north end to the east 270 by 120 ft. From this elevation to about 900 ft. below the surface, the ore spreads out to the east and south, being about 700 by 500 ft. between these vertical limits. Below this it pinches out and at 1060 ft. below the surface is only 100 by 100 ft. The nearest point of the orebody—which is on the northwest side—is 600 ft. southeast of the Sacramento shaft. The country rock is all porphyry. Fig. 1 is a vertical east-west section showing orebody and original surface.

The surface over the orebody was fairly regular and in this area and adjacent to it were portions of the surface plant, roads, and the main plant railroad to the concentrator. Fig. 2 is a map of the surface over the porphyry area.

Caving operations started in April, 1925, at the approximate top of the orebody. The first lift was on the 600-ft. level; the ore was extracted by means of the Morenci timbered slide method and was from 425 to 455 ft. below the surface over an area of about 22,000 sq. ft. This lift was finished in May, 1926. The second lift, on the 650-ft. level, was already in production by May, 1926, so extraction of ore was continuous. This lift was 48 ft. high and covered an area of about 30,000 sq. ft. The ore from this lift was extracted by means of a modified Morenci timbered slide method. This lift was finished in March, 1927. The direction

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* Mining Engineer, Phelps Dodge Corporation, Copper Queen Branch, Mines Division.

of draw was from south to north. The third lift, or 725-ft. level lift, was started in the latter part of 1926 and was in full production by the time the 650 lift was finished. By April, 1927, the southern end of the 725 lift for 70 ft. toward the north was completely drawn, and drawing was taking place in the next 60 ft. over the width of the block.

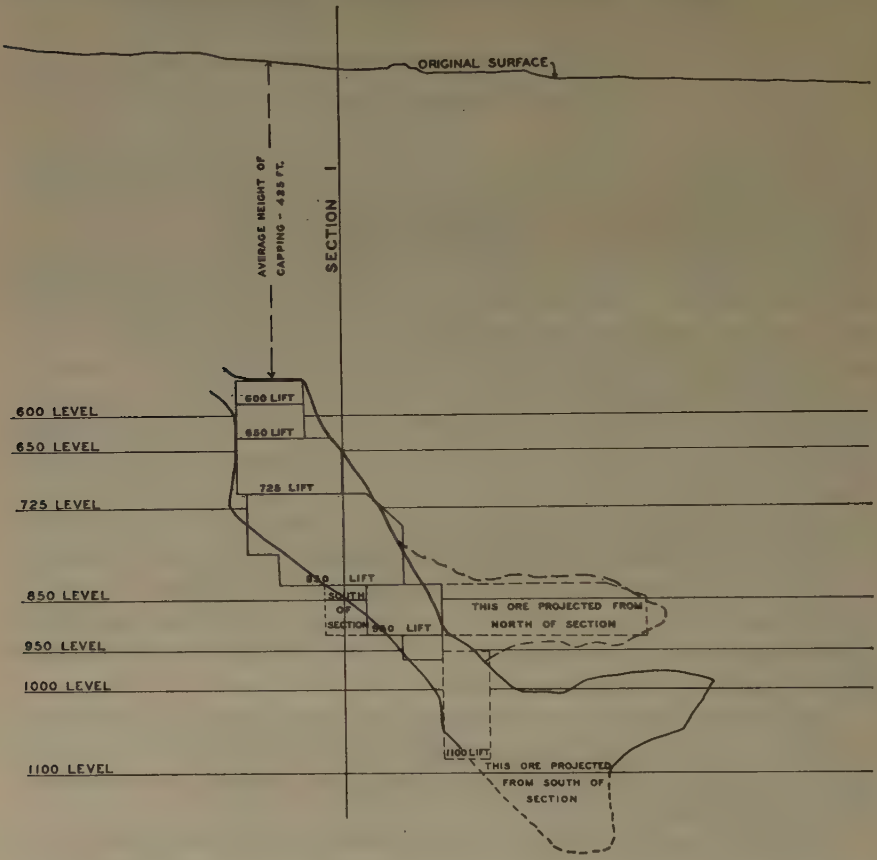

FIG. 1.—SECTION A. ORE AND BLOCK OUTLINES, LOOKING NORTH.

During 1926, several lines of elevations with fixed points at stated intervals were run and a careful survey of the surface was made. This was done as a matter of protection for the plant buildings and plant railroad. Elevations were taken on these several lines monthly; subsidence was noted and at the same time a careful survey was made to note any surface cracking.

In April, 1927, two years after caving started, there was a maximum subsidence of 6 ft. and subsidence had taken place over an area of 199,000 sq. ft. The volume of mining to April, 1927, was 3,325,000 cu. ft. The volume of subsidence was 523,000 cu. ft. This represents a volume of 6.358 cu. ft. mined to 1 cu. ft. of subsidence. The area of horizontal projection of caving blocks was 43,000 sq. ft. Cracking was pronounced at this time and assumed an oval shape with the long axis east and west. Figs. 3 and 4 are vertical sections through the caving blocks showing amount of subsidence, outside cracks, angles of break and subsidence as of April, 1927.

During the remainder of the year 1927, all the caving was from the 725-ft. lift. This lift was 75 ft. high and covered an area of about 52,000 sq. ft. There were two separate blocks on this level, with a horse of waste between them. Here the direction of draw was changed. The direction of draw of the main block was from south to north, as on the lifts above, but the north block was started at almost the same time as the main block. Considerable water was encountered on the north end of the orebody and in order to keep as much of the ore as dry as possible a north block was started at the same time as the block at the south end, which aided materially in keeping all but the north blocks fairly dry.

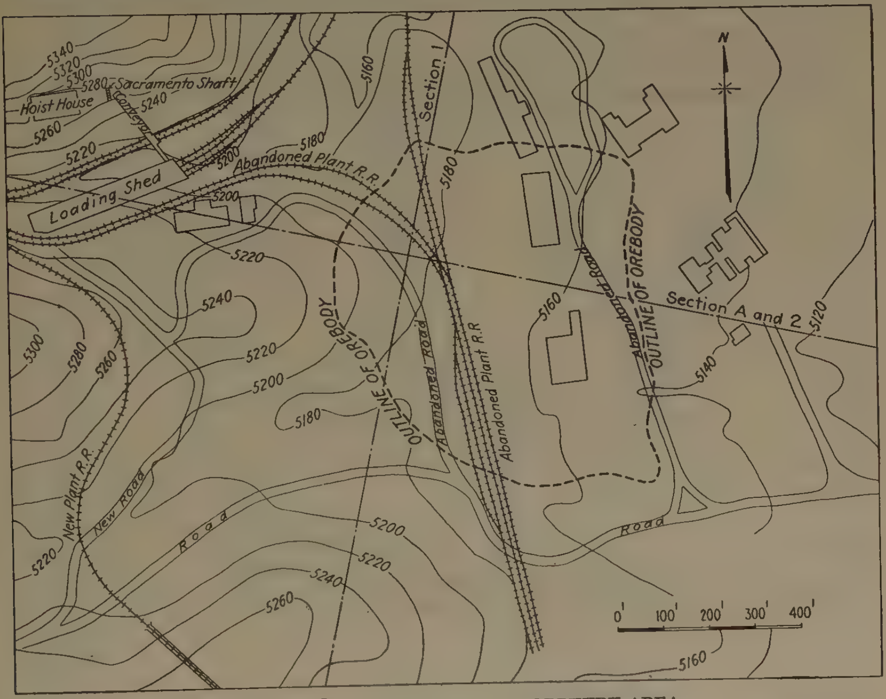

FIG. 2.—MAP OF SURFACE OF PORPHYRY AREA.

The fourth lift, or 850 lift, was started in January, 1928, and by April, 1928, when the 725 lift was finished, the 850 lift was in full production and about 200,000 cu. ft. had been extracted from the southern end of the lift.

At that time, three years after commencement of operations, three lifts were finished. A vertical depth of 153 ft. had been caved and the area of stoping enlarged. The surface showed considerable subsidence, and cracking had extended outward. The maximum subsidence was 17 ft. and the area over which subsidence had taken place had extended to 379,000 sq. ft. The volume of mining to April, 1928, was 7,095,000 cu. ft. and volume of subsidence was 1,749,000 cu. ft. This represents a volume of 4.057 cu. ft. mined to 1 cu. ft. of subsidence. The area of horizontal projection of caving blocks was 59,200 square feet.

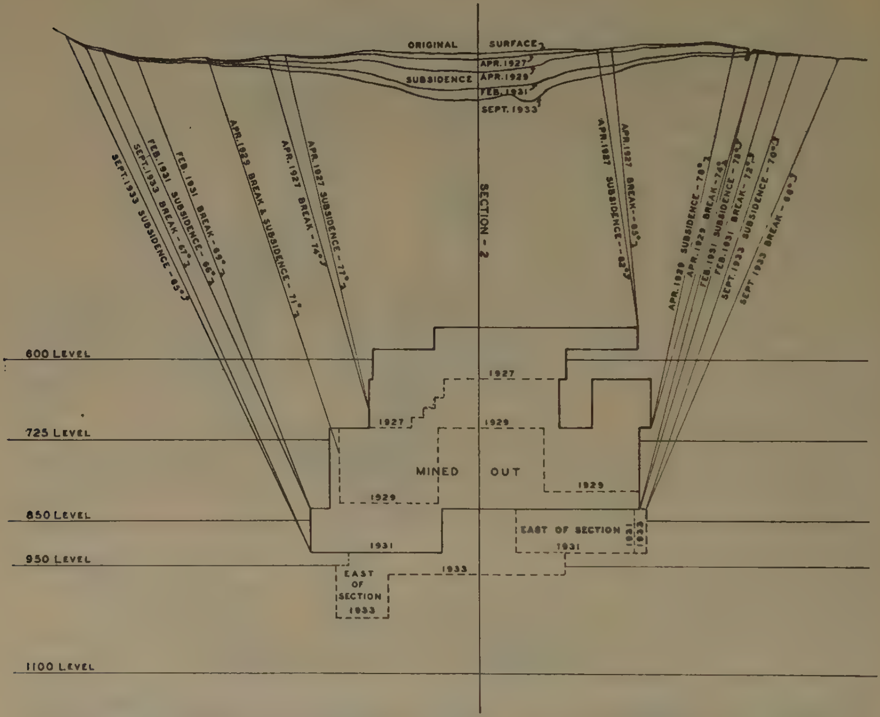

FIG. 3.—SECTION 1. LOOKING WEST. SHOWING BREAK, SUBSIDENCE AND MINING AS OF APRIL, 1927 AND 1929, FEBRUARY, 1931 AND SEPTEMBER, 1933.

During the remainder of 1928 and until April, 1929, caving operations were confined entirely to the 850 lift. This lift is 120 ft. high and has an area of approximately 80,000 cu. ft. The 850 lift was somewhat southeast of 725 lift. The method of extracting the ore on this lift was by pony sets and finger raises.

In April, 1929, four years after operations were started, the southern quarter of the 850 lift was 97 per cent drawn out, the western quarter was 35 per cent completed and the northern quarter, or wet block, had 80 per cent of the ore extracted, while the eastern quarter was just starting.

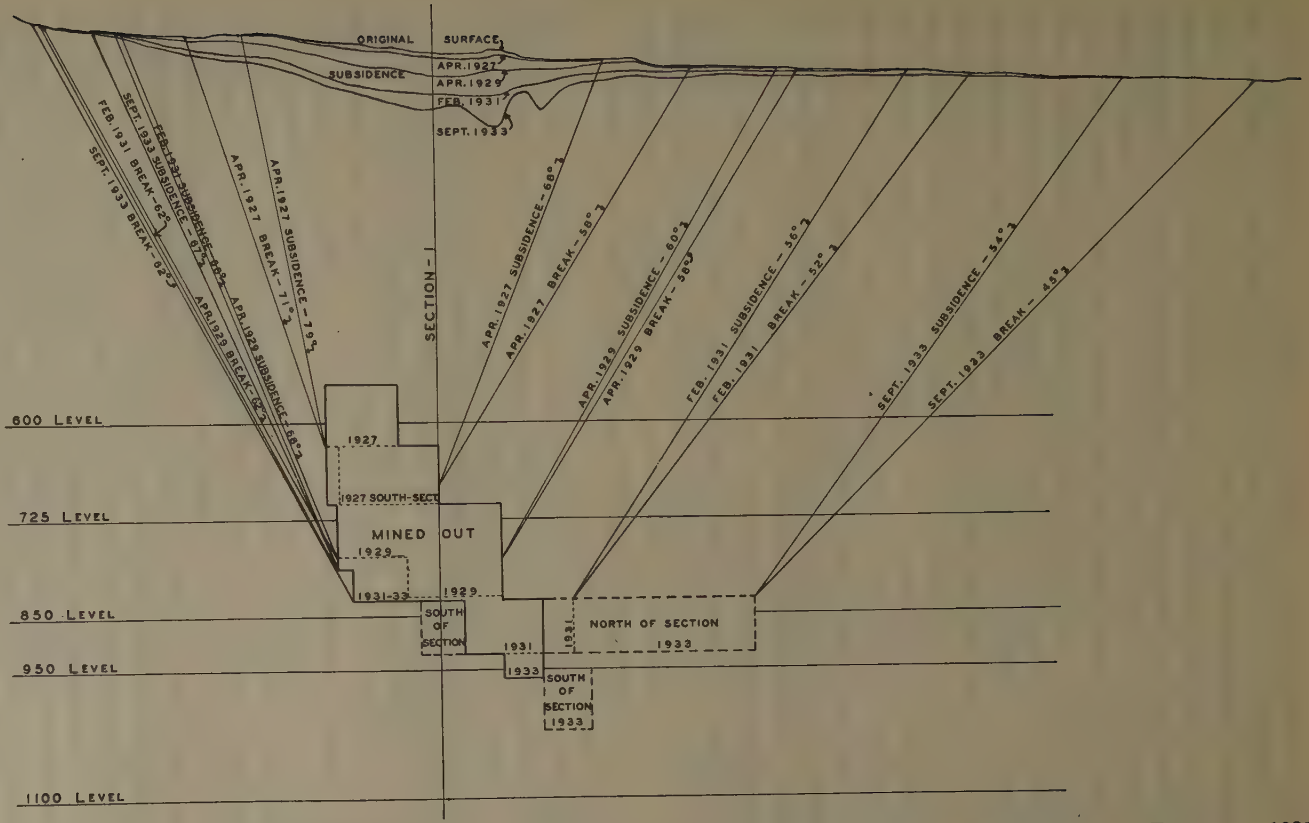

Fig. 4.—Section 2. Looking north. Showing break, subsidence and mining as of April, 1927 and 1929, February, 1931 and September, 1933.

By that time, the movement of the surface was noticeable and roads and railroads were rerouted in order to be perfectly safe. Cracking had progressed materially to the east, while the cracks on the other three sides were much more numerous and definitely larger. The maximum subsidence at this time was 28 ft. and the area over which subsidence had occurred was approximately 695,000 sq. ft. The volume of mining to April, 1929, was 10,812,500 cu. ft. and volume of subsidence was 3,365,700 cu. ft. This represents a ratio of 3.213 cu. ft. mined to 1 cu. ft. subsided. The area of horizontal projection of caving blocks was 77,050 sq. ft. Figs. 3 and 4 show the progress of mining, the subsidence, outside cracking, and the angles of break and subsidence as of April, 1929.

A year later (April, 1930), the southern quarter of the 850 lift was finished, the western quarter was 91 per cent completed, the northern quarter lacked 5 per cent of being drawn out and the eastern quarter had 70 per cent extracted. On the surface the cracks were gradually extending in all directions and becoming larger, and several holes had opened. The maximum subsidence at this time was 40 ft. and the area over which subsidence was noted had extended to about 950,000 sq. ft. The volume of mining in the caving blocks to April, 1930, was 16,150,000 cu. ft. and the corresponding volume of subsidence was 6,550,000 cu. ft. This gives a ratio of 2.466 cu. ft. mined to 1 cu. ft. subsidence. The area of horizontal projection of caving blocks had increased to 98,300 square feet.

In the meantime as the 850 lift was nearing completion, the 950 lift was being prepared; in April, 1930, drawing began in the north block, and a month later drawing started in the south block. The 950 lift was 70 ft. below the 850 lift and slightly to the south and east of it.

In February, 1931, extraction had progressed to the extent that the 850 lift was exhausted and 95 per cent of the 950 north block was drawn out The 950 south block was 70 per cent completed and the 950 central block was about one-third depleted. The two eastern blocks were just in the preparatory stage. By this time, the surface had become greatly disturbed; the holes previously showing had increased in depth and circumference and new ones were beginning to appear. Cracking of the surface had greatly increased and expanded. The maximum subsidence in February, 1931, was 65 ft., in two of the holes—outside of the holes the maximum subsidence was 55 ft. The approximate area over which subsidence was observed had reached 1,220,000 sq. ft. The volume of ore mined in these caving blocks was 22,438,500 cu. ft. and the corresponding volume of subsidence, 10,515,000 cu. ft. This represents a ratio of 2.134 cu. ft. mined to 1 cu. ft. subsided. The area of horizontal projection of caving blocks was now 114,700 sq. ft. Figs. 3 and 4 show the progress of mining, the subsidence, the outside cracks and the angles of break and subsidence as of February, 1931.

During 1931, the 950 lift was exhausted, 35 ft. of ore beneath the 950 central and south blocks were drawn out and the 1100 lift started with a block 70 by 85 ft., which was east of the 950 south block. The 1100 lift was on a horizon 160 ft. below the 950 lift. A total of 420,000 cu. ft. was drawn from this 1100 block. Caving operations ceased on Jan. 1, 1932.

On Jan. 1, 1932, the surface deformation had progressed rapidly; old holes were gradually deepening and new holes were appearing, cracking had extended materially outward, especially to the east, and the maximum subsidence was 85 ft. This was in two of the holes; outside of the holes the maximum subsidence was 70 ft. The area over which subsidence had been observed was approximately 1,500,000 sq. ft. The volume of ore mined in these caving blocks was 28,250,000 cu. ft. and the corresponding subsidence volume was 14,614,500 cu. ft. This gives a ratio of 1.933 cu. ft. mined to 1 cu. ft. subsided. The area of the horizontal projection of the caving blocks had increased to 161,000 square feet.

After the cessation of caving, subsidence continued, gradually slowing up so that after 20 months the movement of the ground is very slight and it seems probable that two years after mining ended the subsidence will have reached a maximum.

In September, 1933, the maximum subsidence had reached 93 ft. in a couple of the holes and outside of the holes it was 78 ft. The area over which subsidence had been observed was approximately 1,535,000 sq. ft. The total volume of subsidence at this time was 15,855,600 cu. ft., which gives a ratio of 1.78 cu. ft. mined to 1 cu. ft. subsided. Fig. 5 shows the outside cracks and the contours of subsidence and their relation to the caving blocks and the Sacramento shaft. Figs. 3 and 4 are vertical sections showing the mined-out caving blocks, the subsidence, outside cracks, and the angles of break and subsidence as of September, 1933. The farthest disturbance of surface is to the east and represents an angle of break of 45°. To the north and northwest, a major fault steepened this angle and for a long period no cracking beyond this fault was observed. To the south and southwest, the surface rises rather abruptly and other underground workings also affect this area.

The footwall of the orebody is on the west, and the rock of the footwall is more highly siliceous, harder and denser than the hanging wall on the east. The hanging-wall rock in many cases is extremely soft and friable and this is probably responsible for the extreme extension of cracking in this direction and for subsidence extending so far to the east.

Omitting the small block on the 1100 lift, the total height of caving blocks mined out was 385 ft. and the total subsidence 78 ft. The area of horizontal projection of blocks is 161,000 sq. ft. and the area of ground subsided is 1,535,000 square feet.

Table 1 shows at a glance the area of horizontal projection of caving blocks, volume mined from blocks, area over which subsidence occurred, volume of subsidence, ratio of volume mined to volume subsided, total height of mining, and maximum subsidence at approximately yearly intervals. Table 2 shows the angles of break and angles of subsidence on the separate sections over yearly periods.

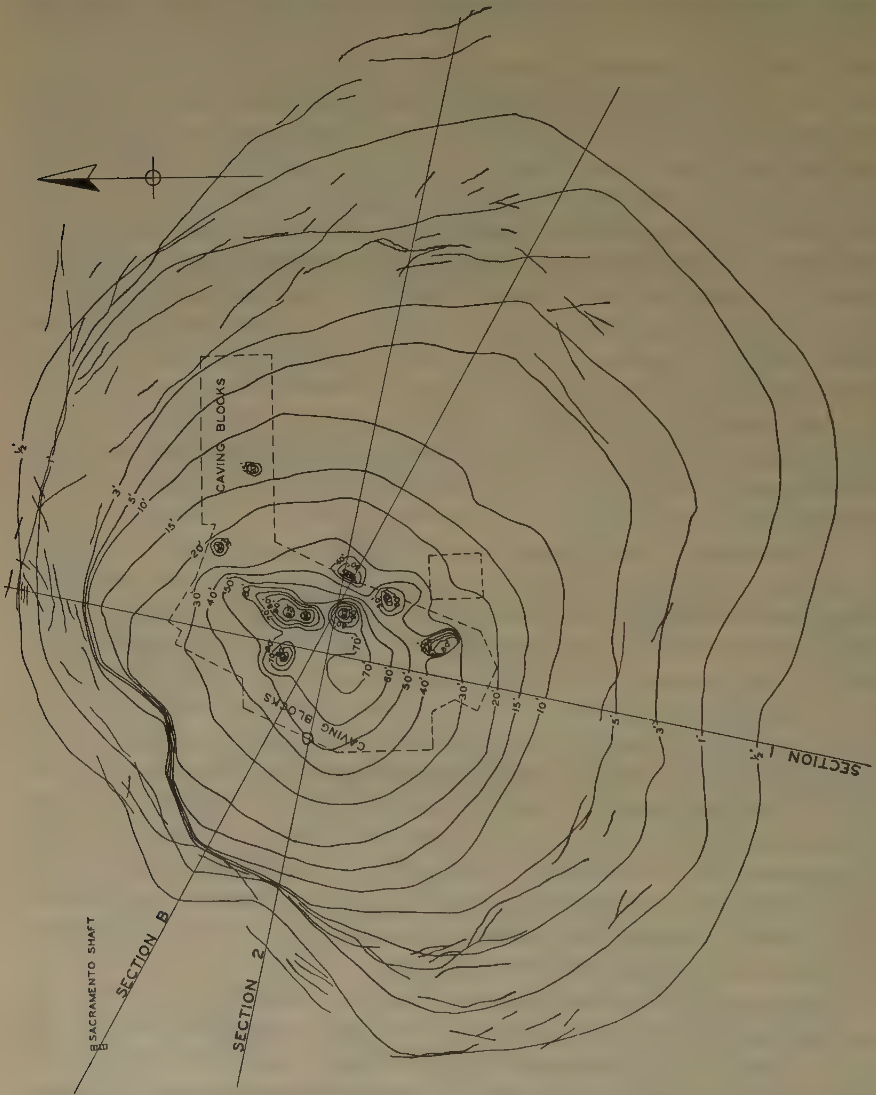

FIG. 5.—LIMIT OF CRACKING AND CONTOURS OF SUBSIDENCE AS OF SEPTEMBER, 1933.

The original amount of capping over the top of ore was 425 ft. thick. The top ore area was only about one-fourth the area of the largest lift, so the actual amount of capping to break down was more than 425 ft.

thick. The vertical height of ore mined was 385 ft. and the maximum subsidence—except in holes—was 78 ft., or slightly more than one-fifth the height of ore mined out. The first subsidence noted was during the latter stages of draw from the second lift, or after approximately 1,800,000 cu. ft. of ore had been mined and a vertical height of ore mined was around 70 ft. Over a year and a half elapsed after caving operations started until subsidence was noted.

TABLE 1.—*Mining and Subsidence Data*

Date	Area, Horizontal Projection Caving Blocks, Sq. Ft.	Volume Mined from Caving Blocks Cu. Ft.	Area Surface Subsided, Sq. Ft.	Volume Surface Subsidence, Cu. Ft.	Ratio Volume Mined to Volume Subsidence	Total Height Mined, Ft.	Maximum Subsidence, Ft.
April, 1927.....	43,000	3,325,000	199,000	523,000	6.358	80	6
April, 1928.....	59,200	7,095,000	379,000	1,749,000	4.057	153	17
April, 1929.....	77,000	10,812,500	695,000	3,365,700	3.213	196	28
April, 1930.....	98,300	16,150,000	950,000	6,550,000	2.466	256	40
February, 1931.	114,700	22,438,500	1,220,000	10,515,000	2.134	315	55
January, 1932..	161,000[a]	28,250,000	1,500,000	14,614,500	1.933	385	70
September, 1933	161,000[a]	28,250,000	1,535,000	15,855,600	1.78	385	78

[a] Of this area, 39,000 sq. ft. of this 161,000 sq. ft. has a height of only 70 to 78 ft. and is east of the regular column, so that for the 385 ft. total height there is only an area of 122,000 square feet.

TABLE 2.—*Angles of Break and Subsidence*

Date	North-South Sections				East-West Sections			
	South of Blocks		North of Blocks		West of Blocks		East of Blocks	
	Angles of Break, Deg.	Angles of Sub., Deg.	Angles of Break, Deg.	Angles of Sub., Deg.	Angles of Break, Deg.	Angles of Sub., Deg.	Angles of Break, Deg.	Angles of Sub., Deg.
April, 1927............	74	77	Inverse −85	Inverse −82	71	79	58	68
April, 1928............	73	73	72	88	62	72	56	64
April, 1929............	71	71	74	78	62	68	58	60
April, 1930............	69	66	71.5	75	62	68	54	57
February, 1931........	69	66	72	75	62	68	52	56
January, 1932.........	68	66	68	72	62	68	48	57
September, 1933.......	67	65	66	70	62	67	45	54

The only irregularity of surface, with the exception of the 10 holes, caused by this large subsidence was on the north and northwest side of the subsided area. This was along the major fault and here the escarpment varied from 8 to 20 ft. A definite break was observed early in caving operations on this fault.

When caving operations started it was presumed that eventually as the mining increased in depth the Sacramento shaft would be affected

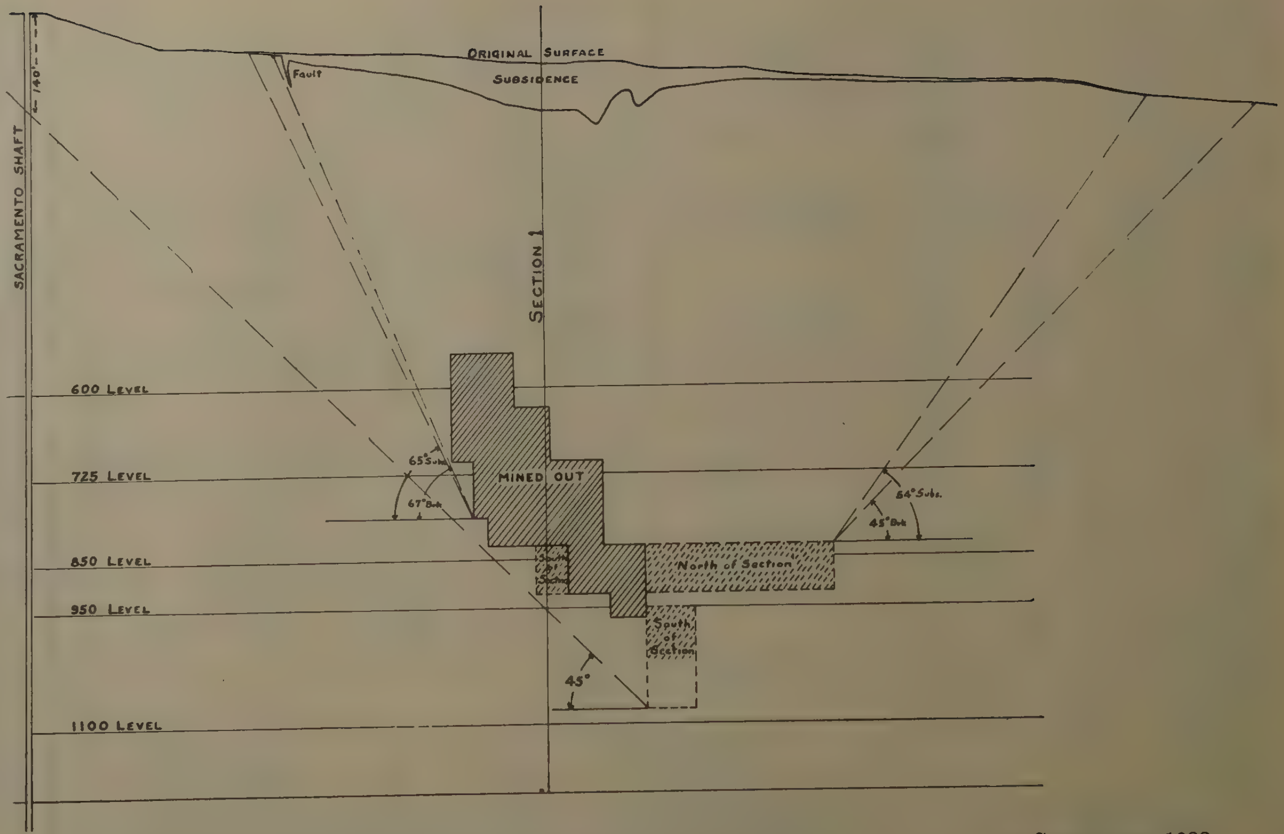

FIG. 6.—SECTION B. LOOKING NORTHEAST, SHOWING BREAK, SUBSIDENCE AND MINING AS OF SEPTEMBER, 1933.

by the breaking up of the ground and subsidence and would have to be abandoned. Sites for a new porphyry shaft were looked over and a site practically agreed upon. This was another reason the subsidence and cracking up of the surface was watched so carefully. An angle of 45° for breaking was used and it was seen that this line cut the shaft from 110 to 160 ft. below the collar. If conditions had not been affected by a major fault we find that the angle of 45° was about right, as that is the final angle of break on the east side of the caving blocks. However, conditions were not regular and a major fault on the northwest side of the caving blocks halted the breaking and subsidence and this very fortunate occurrence of a natural barrier saved the cost and trouble of a new hoisting installation. From Fig. 6 we see that the angle of subsidence and angle of break in this direction in September, 1933, were 65° and 67° respectively. Also, the extreme break is only 20 ft. beyond the major fault and the edge of subsidence is only 50 ft. beyond the fault.

From all the above data it can be seen that faults and previous mining at lower horizons have a direct effect on breaking and subsidence of the surface over mined-out orebodies. We hope that the data contained in this paper will give a clear idea of the actual chain of events as they occurred here and may be a benefit in a general way in future operation of a similar character.

DISCUSSION

(*George S. Rice presiding*)

[This discussion refers also to the paper by Mr. Trischka, which begins on page 173.]

MEMBER.—Do natural caves occur in the limestone?

G. SHERMAN,* New York, N. Y.—There were caves in the limestone, often above ore deposits and chiefly in the southwestern part of the district where the ore horizon is near the surface. One was so large that a dinner was served in it for a meeting of the A.I.M.E. when Dr. James Douglas was president. A small cave was mined out and is now set up in the American Museum of Natural History. The larger ones have collapsed, probably from ground movement.

G. S. RICE,† Washington, D. C.—Were there valuable surface improvements in this area or could it subside without damage?

G. SHERMAN.—There were three shafts within the influence of mining in this block, of which only two are shown, the Czar and the Holbrook. The ore deposit was comparatively flat and not far below the surface and there was so much soft ground caused by the alteration and decomposition of the limestone that it was very difficult to maintain long lines of transportation. That is why shafts were sunk closer together than usual. The Holbrook shaft was finally lost, and the main break on the southwest boundary of the block passed through the Spray shaft and engine foundation and bent one of the legs of the steel headframe.

R. E. TALLY, ‡New York, N. Y.—Are the ores pretty well worked out?

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* Consulting Engineer.

† Chief Mining Engineer, U. S. Bureau of Mines.

‡ Vice President, United Verde Copper Co.

G. Sherman.—Yes, they are pretty nearly gone in this area. The lower limit of the ore, so far as we know, is the bottom of the Cambrian limestone, which pitches to the southeast. The Czar shaft is 400 ft. deep, the Holbrook 600, and the Spray 800, corresponding roughly to the pitch.

R. E. Tally.—There is more or less natural subsidence due to the extraction of the ore and breaking to the lines of weakness?

G. Sherman.—Yes, planes of weakness govern the limits of subsidence, but not always major faults, unless they approach the natural break line. There is an example of the latter case a little to the east of Mr. Trischka's block. An orebody lies against the Dividend fault, which has a throw of 2000 ft. or more, and beneath Mule Creek, a highway, a street railway, sewer system, etc. The ore and capping were so soft and wet that it was almost impossible to mine by square setting. In order to slice, it was necessary to divert the creek and push the highway and other improvements back on the footwall of the fault.

The ore, which extended from the 200 level to below the 400, was top-sliced down. The capping slid down on the fault, which has a pitch of about 65°, without breaking into the footwall. The subsidence started soon after mining began but was progressive and probably still continues.

The big crack on the southwest boundary is plainly traceable. The break on the Ella fault to the southeast is less distinct, as stoping continued in that direction and the ground is more uniformly broken. The principal difference in conditions was the increasing thickness of barren limestone over the ore.

G. S. Rice.—As a general rule, the breaks start vertically from the surface but presumably curve to a flatter angle in going downward. Or, starting at the excavation the break in homogeneous rock goes up at an angle through the solid formation, then as it nears the surface a slide like a landslide begins and the final break at the surface is more or less vertical.

G. Sherman.—Yes. The break goes down perhaps following a joint in the limestone and then offsets on the bedding.

R. E. Tally.—Not necessarily vertical?

G. Sherman.—Vertical for short distances and then jogging off. The tendency is affected by local disturbances. There might be two planes of weakness not in the same line. They would both open up with adjustment cracks between.

R. E. Tally.—I should say not necessarily continuous.

G. Sherman.—They are not all continuous.

G. S. Rice.—The effect, I think, would be much the same as it is at Miami, which we had described in former papers. It was very evident there that the final breaks at the surface were vertical. The angle of draw might have been as flat in one case as 35°, the ground riding on a fault plane, but when it came near the top, surface blocks moved laterally. It was virtually a landslide. The blocks are carried on top of the moving ground, moving toward the center.

R. E. Tally.—Does Mr Kantner's paper refer to the Sacramento Hill porphyry ore?

G. Sherman.—To a portion of one of the two orebodies of the Sacramento Hill porphyry. The west orebody was mined by open-pit shoveling. The section referred to in this paper is a small and richer extension from the east orebody, to the southeast, of which a considerable part has been taken out.

G. S. Rice.—The subsidence represents present conditions, does it not?

G. Sherman.—It represents conditions on the dates given. I believe there has not been much change since the last dated section.

G. S. Rice.—The change now continues?

G. Sherman.—Yes, but subsidence is much slower. While it may continue for a long time the rate will decrease progressively. The author thinks it will end in about two years after mining stopped, but I am a little doubtful. The capping in place occupied much less space. It expanded in caving and is now under pressure from its own weight.

G. S. Rice.—There were no faults?

G. Sherman.—There were no major faults. The porphyry was not very hard and very closely cut by fissuring, which explains the symmetry of the area of subsidence over the block mined. The steeper angle of break on the Sacramento shaft side was probably caused by harder underlying porphyry than a fault of greater throw.

G. S. Rice.—Are the small areas marked on the map actually holes?

G. Sherman.—They are holes, but only open for a short distance down, say 20 ft. or so. A hole in the soft capping would not stay open for any great depth.

G. S. Rice.—You referred to the angle of break.

G. Sherman.—There are two points on the surface through which the break could be drawn. One is the point where well marked subsidence begins, and the other is the extreme limit of fine open cracks with little, if any, vertical displacement.

The steepest angle to the point of marked subsidence was 70°, and the limit of visible cracks was 66 in the same section. In the flattest section, the corresponding angles were 54° and 45°.

G. S. Rice.—Mr. Kantner's paper is particularly valuable to the Committee, in that it is one of the few instances of papers that give definite volumetric figures of excavation and volume of ground subsided, through having accurate surveys made in advance of subsidence. It will be interesting to find in future to what extent subsidence does continue or whether there is a gradual filling of the broken ground by infiltration of mineral-carrying waters.

L. Eaton,* New York, N. Y.—In these papers by Mr. Trischka and Mr. Kantner, two things struck me as very interesting. One is the relative area of subsidence and of ore mined, and the other is the relation of the volume of subsidence to the volume of ore mined.

Some years ago I did a good deal of work along these lines in the Lake Superior district, which probably has the largest number of examples of subsidence in the United States, because the caving systems of mining, top slicing and sublevel caving are so largely used; and there I got some figures, but they did not check up very well. On the Gogebic Range in one of my first determinations I found a subsidence on surface of 1 cu. ft. for every 3 cu. ft. mined and in an old mine at Ishpeming, Mich., there was a ratio of almost one to one; that is, about 1 cu. ft. of subsidence to 1.1 cu. ft. of excavation. That subsidence had been going on for many years and was practically complete. Other ratios were between these limits.

The relation of the depth of the subsidence to that of the excavation varies widely also. Under Lake Angeline in Ishpeming, Mich., there were deposits of ore, from which over 20 million tons of ore were mined, and to get this ore it was necessary to pump out the lake. The outline of the area that subsided followed closely that of the ore mined, and the depth of the subsidence in places was over 150 ft. When the lake was pumped out 40 ft. of mud was left in the bottom, and much of this ran into the mine at various times so that the apparent subsidence was greater than the actual.

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* Consulting Mining Engineer.

There is another mine in Ishpeming, which forms an interesting contrast in that the hanging wall is quartzite, which is broken into large blocks by minor faults. These may be 200 or 300 ft. long and 100 or 200 ft. wide. The ground does not break short, as jasper does. In jasper we found that the areas of subsidence followed closely those of mining, whereas in the stronger rocks like quartzite, the subsided area is much larger. In the mine that I spoke of over 6 million tons of ore had been mined from an area of only 3 or 4 acres, and the total subsidence covered over 20 acres. Here we had an example similar to the one that Mr. Sherman showed, in that the angle of break on the hanging-wall side, which was across the formation, was flatter than the angle of break on the footwall side, which was more or less parallel to the contact. I have found by experience in other places that this is very often the case. It is not always true, however, but within reasonable limits, if the dip of the footwall is somewhere near that of the theoretical angle of break, the break will follow the footwall, whereas if the break is across the formation it will be at a much flatter angle. In the mine that I spoke of the break on the footwall side was at an angle of 60° to 70°, whereas the angle was 35° on the hanging-wall side.

G. S. Rice.—In some instances of excavation, deep but of limited width, I have doubted whether the broken rock filling will ever be compressed back to a one to one density. That is, relatively hard rocks when broken perhaps occupy one and one-half times the original volume, certainly one and a quarter times. That being so, would the broken rock filling be crushed back to the usual density of the solid rock?

L. Eaton.—I cannot explain the ratio of one to one either; but that is the result that I got from my calculations. I had only the old contours and records of production to go by. That is why I said that the figures were not as conclusive as I should have liked to have them. As for broken rock taking up more room than it did in place, I think that we are likely to overestimate the increase. I remember an instance at the Lake mine, where a stope caved in. It really was not much more than a drift, but it caved in and caught two men; and within half an hour after the cave occurred the caved ground would stand open without support for 3½ ft. in front of timber. It was just as solid as the ground that we had been mining.

G. Sherman.—I think the nature of the capping, and the height and shape of the volume of subsidence have much to do with the density of its final state. This orebody was formed near the upper surface of the porphyry intrusion. It was prepared for receiving the mineralizing solutions by cooling, shrinkage and brecciation, and those agencies extended for some distance into the capping. The close uniform pattern of the planes of weakness and the comparatively soft nature of the rock accounts for the gradual sinking and the symmetrical area of subsidence, and eventually will result in compact settlement. If the fragments were large, angular and strong they might never in our time make a dense mass.

Subsidence Resulting from the Athens System of Mining at Negaunee, Michigan

By Charles W. Allen,* Ishpeming, Mich.

(New York Meeting, February, 1933)

The Athens mine is in the city of Negaunee, Mich. It is one of the larger producers of soft hematite ore on the Marquette iron range. The property is held under lease by the Athens Iron Mining Co., and is operated by the Cleveland-Cliffs Iron Co., which is a partner with Pickands, Mather & Co. in the Athens company. The mine was opened in 1913, and shipments of ore commenced in 1918, the total reaching 3,063,711 tons up to Jan. 1, 1932.

Diamond-drilling from surface, together with a knowledge of geological conditions in neighboring properties, gave a comprehensive picture of the location, size and shape of the orebody before shaft-sinking started. Striking east and west, the orebody entered the east boundary of the property at a depth of about 1500 ft. below surface. A pitch of 15° to the west carried it down to about 2500 ft. on the west boundary, about double the depth of that of any soft-ore property heretofore worked on the Marquette Range. The width of ore averaged close to 500 ft., and the thickness 300 feet.

The depth of the deposit and its size, together with an overburden of quicksand, which in places reached a thickness of 150 ft., presented unusual mining problems. The use of a caving system made surface drainage into the orebody inevitable, because the north boundary of the deposit was a vertical diorite dike 200 ft. wide, and the south boundary a northward-dipping impervious slate. S. R. Elliott, manager of the mining department of the Cleveland-Cliffs Iron Co., who was superintendent of the mine at that time, conceived the idea of starting at the bottom of the orebody and working upward, reversing the usual mining procedure. This innovation divided the Athens orebody into a series of steps, or blocks, the first of which was adjacent to the western boundary, and therefore in the lowest portion of the deposit. Top-slicing was to start under the capping in this first block by raising from the bottom level, and development and mining were to progress upward in successive blocks to the east. A complete description of the Athens system of mining was

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* Mining Engineer.

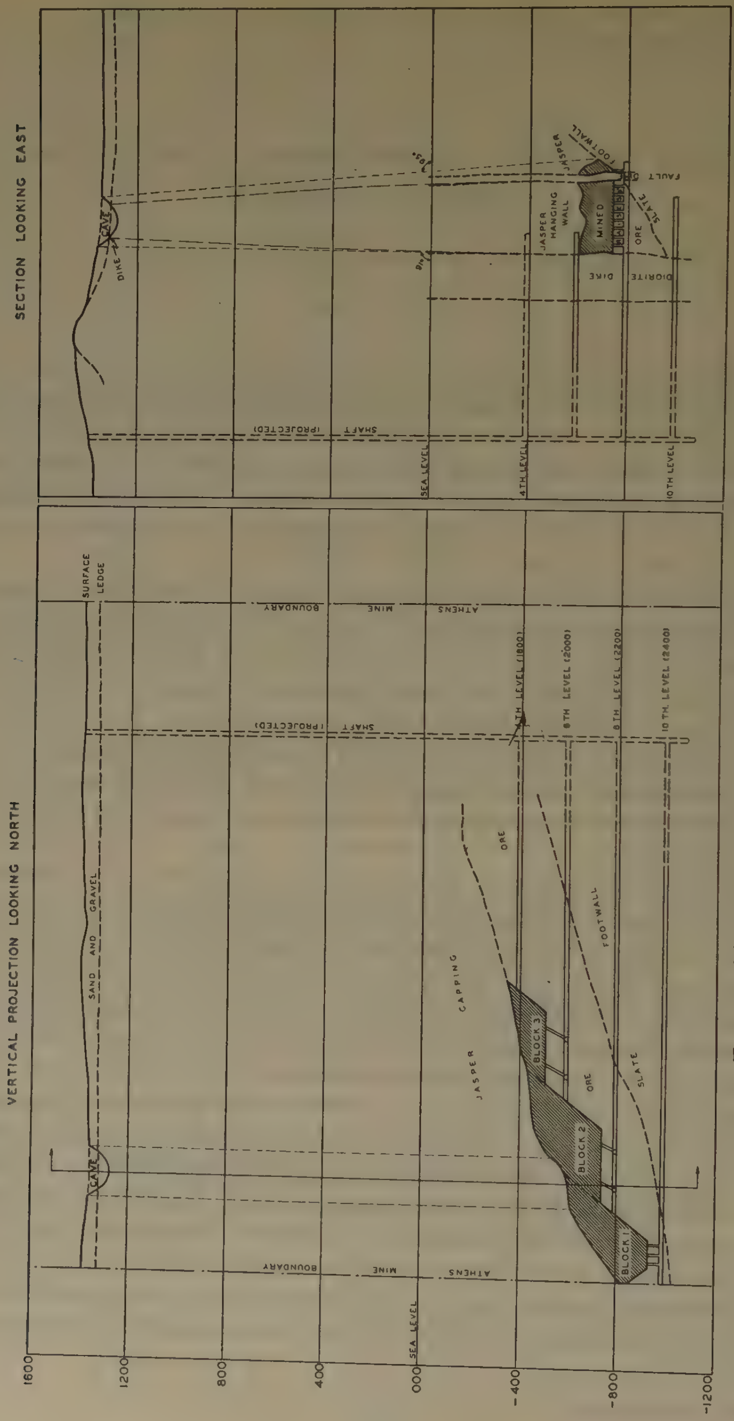

FIG. 1.—MINING SYSTEM AND RESULTING SUBSIDENCE, ATHENS MINE.

published in the TRANSACTIONS in 1921.[1] The results expected from it were the delaying of a cave to surface and the location of this eventual cave over a mined area below the working places. In this way, prohibitive pressures would be avoided, providing a saving in timber cost; dry working places would increase efficiency and allow a saving in freight in shipping a dry ore; a decreased pumping expense would result from the longer length of time required for the cave to reach surface; and the increased safety in placing the eventual surface cave over an abandoned portion of the workings. The principal argument against the use of the system was the increased interest on the investment arising from delayed production.

A shaft site was selected some distance north of the orebody, and the shaft sunk to its ultimate depth, a distance of 2489 ft. The shaft construction was the Cleveland-Cliffs Iron Company's standard vertical, four-compartment, concrete shaft, equipped throughout with steel sets. A connection was made to the Negaunee mine by a drift 1100 ft. below surface, and development work was started on the two bottom levels, the eighth at a depth of 2200 ft., and the tenth at a depth of 2400 ft. When the orebody was reached, its outline was determined, and the production of ore started from block No. 1 at the western and lower end of the mine in 1918.

A length of 300 ft. along the strike was decided upon for the first block, and this same distance was later used in blocks 2 and 3, because with two crosscuts per block it afforded an economical and efficient distance for the use of scrapers in mining. This 300-ft. length caused the first production of ore to issue from above the eighth level instead of the bottom (tenth) level, because of the steepness of the capping in this section. The mining limit on the east side of the block was inclined at an angle of 55° to hold in place the ore to be mined in block 2. A considerable amount of water was encountered in developing this first block at the lower end of the deposit. This was not so in the development of blocks 2 and 3, and for the five years preceding the surface cave the amount of water handled was approximately 225 gal. per minute, most of which issued from the then worked-out block 1.

Development work on the sixth level at a depth of 2000 ft. below surface, and the fourth level at a depth of 1800 ft., progressed while mining operations were under way in the first block. The westward retreat of the mining limit and the northward dip of the slate footwall steadily reduced the size of this block, and in 1921 the second block was started under the capping at a depth about midway between the fourth and sixth levels. Development work on the fourth level disclosed a local flattening in the slate footwall in the section south of what would later be block 4, and as it was desired to increase production at that time, two small blocks were opened and mined so that a more nearly uniform width of ore would be secured as the schedule of blocks progressed upward to the east. At the time when maximum production from block 2 was reached, the encroaching mining limit and slate footwall had so reduced the area of block 1 that it was abandoned a short distance above the bottom level. Block 3 was then started under the capping at the fourth level elevation in 1926, and in a year or two a well balanced production was coming from blocks 2 and 3.

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[1] S. R. Elliott: Athens System of Mining. *Trans.* A.I.M.E. (1921) **66**, 220.

About this time the advantages of the Athens system of mining began to show themselves. Ideal mining conditions prevailed in these blocks. Because of the small amount of water, the Athens was the first mine in the district to become 100 per cent mechanized through the use of scrapers. The tons per man per day increased, and the cost decreased. The use of scrapers permitted rapid extraction and this kept timber costs at a minimum on the mining sublevels. The ore has a tendency to swell when newly developed, and this, combined with great pressure and the softness of the ore, make rapid extraction necessary for highest efficiency. The moisture content of the ore placed on stockpile has been steadily decreasing during the past few years, and a higher iron content (dry analysis) is indicated. The latter is due to improvements in the top-slicing system of mining, a description of which follows: The arrangement of main-level crosscuts and raises divides each block into pillars 35 by 150 ft. on the sublevels, the pillars being at right angles to the crosscuts with the raises in the center. One drift and a slice on each side of the drift are driven 75 ft. in length on each side of the raise. The sublevel intervals are 11 ft., using an 8-ft. leg, and 12 ft., using a 9-ft. leg. Accidents due to falls of ground have been reduced, because these intervals leave no ore between the floor of one sublevel and the top of the timbering on the sublevel below. Lagging is used where necessary on the sides and back. The floor is covered down with 9½-ft. poles nailed to three crosspieces. Under the rock capping, or near areas of lean ore, the poles are laid close together, but where the timber mat is sufficiently thick they are spaced from 6 to 12 in. apart. All open spaces between drift and slices are carefully cross-lagged. This system of covering down prevents runs of rock or lean ore from the back, thus giving a cleaner and more complete extraction.

In June, 1932, mining in block 2 had reached an elevation midway between the fourth and sixth levels, and in block 3 the working sublevel was about 40 ft. above the eighth level. No water had been encountered in the working places in block 2, which were adjacent and under the upper portion of the mat of the abandoned block 1 because of the westward inclination of the mining limit. However, within block 2 there was a good chance for comparison of conditions that probably would have been widespread if the Athens system of mining had not been adopted. About 350 ft. south of the wide dike that forms the north boundary of the orebody there is a smaller dike 20 to 30 ft. thick, known underground as the "fault dike," because of the horizontal displacement of the ore on either side. Between this dike and the slate footwall to the south is a small width of ore in block 2, that was not tapped in block 1, and within this crotch, which represents the whole orebody on a smaller scale, were the only wet working places in the mine, with their inefficient recovery of ore at higher cost.

The ore mined from block 1 represented an excavation roughly 250 ft. deep and 250 ft. wide, the center of which was 2100 ft. below surface. In block 2 the depth was 250 ft., the width 350 ft. and the center 1900 ft. below surface. The combined length was 600 ft. The smallest vertical depth of mining, less than 200 ft., was at the center of this distance because of the inclination of the capping and the mining limit between the blocks. Yet it was over this point that the center of a surface cave appeared on the morning of June 19, 1932. The subsidence of a block of jasper capping nearly 1900 ft. thick resulted from a vertical opening of about $1/10$ that height; convincing proof that subsidence theories must largely take into account existing geological conditions.

Before an explanation of the reason for this unusual subsidence is offered, a description of its effect is given. The immediate shock and pressures were not transmitted through the broken capping and timber mat to the underground workings. The cave-in occurred at 5 o'clock in the morning, during an interval between the pumpmen's change of shift, so there was no one underground at the time. Immediate inspection revealed no evidence of inundation by water or sand, no crushing of drifts or workings, and no trace of the effects that would have resulted had there been an air blast. During the day, however, the water issuing from the workings in block 2 on the south side of the dike above the eighth level and the water from the old block 1 increased in quantity. Preparations were made to take care of any sudden increase, which would raise the quantity above pumping capacity, but this did not materialize. From the first day, the pumping log showed a more or less steady increase from about 300 gal. per minute to a figure about double that quantity, which was reached early in July. Approximately 600 gal. per minute was the average amount pumped for this entire month, and during this period the extent of surface drainage into the cave was indicated by the lowering of the water level about 6 ft. in the surface cave of an idle mine about 1000 ft. to the southwest. The pumping figures for succeeding months show a steady decrease, 540 gal. per minute being pumped in August, 486 in September and 431 in October. When the water table is once more established in the vicinity of the cave, it is thought that the final average figure will show a further decrease, and although the pumping cost per ton of ore produced will show an increase, it will not appreciably affect the operation of the mine.

An unexpected result, because of the location of the cave, is the position of one of the underground sources of the increased issue of water. A course vertically downward would bring the water into the workings between the two dikes in the vicinity of the mining limit between blocks 1 and 2, but the present workings directly under the cave in block 2, which are partly under the timber mat from block 1, have remained dry, as before, while a large share of the increase comes from the previously wet area between the fault dike and the slate footwall. Apparently the cave opened a new water course above the orebody through the fault dike to the impervious footwall, which delivers the increase to the workings below.

An attempted explanation of the reason for the surface cave is based on the assumption that both the north dike and the fault dike extend in a nearly vertical direction from the orebody to ledge. That this is true of the north dike at least is indicated from drill records and the ledge on the north side of the surface cave. The outer portions of both dikes are composed of "soap rock," or fault gouge, and are planes of shearing weakness. This left little or no support for the jasper capping on its north and south borders over the mined area. The jasper capping may then be likened to a wide fixed beam between these planes of weakness, having its own immense weight to support. Assuming that the greatest bending moment was placed over the center of the 600-ft. mined distance, the lower courses of capping were thrown into tension. The tensile strength of the rock in comparison to its weight being safely placed at a lower figure, failure extended upward, and the surface cave resulted.

The capping immediately over the orebody broke into relatively small masses of rock in the arching process above earlier mining operations. This broken rock is thought to have acted later as a cushion above the timber mat for the much larger masses released by failure of successively higher courses in tension. The arch or dome theory can hardly explain the surface subsidence in this instance, because the thickness of capping would seem to be more than enough to fill the opening above the timber mat through enlargement by breakage; and the sudden subsidence of an entire block of the capping south of the big dike seems no more reasonable, for if this had been the case, there was the probability of crushing the underground openings beneath the timber mat.

A study of the surface caves resulting from mining operations at the Maas-Negaunee mines of the Cleveland-Cliffs Iron Co., located about ½ mile north of the Athens mine, reveals striking similarities of occurrence, which may help to forecast future subsidence at the Athens. The Maas-Negaunee orebody is wider but less thick than the Athens deposit. It pitches downward to the west at an angle of about 15°, and mining operations have been conducted for the past few years at a depth below surface of about one-half that of the Athens. The caves at the Maas-Negaunee properties have, without exception, appeared on the line of east-west dikes, all of which are fault planes. The outermost cracks at surface have never been far outside the limits of underground excavations and often have been inside them.

The mining procedure at the Maas-Negaunee is the reverse of that at the Athens, progressing downward from east to west. Since the first cave resulting from the shallower workings, it has been noticeable that the western end of the cave advances by a series of breaks, forming terraces or steps, while the north and south borders follow the lines of dikes. These breaks appear first in the form of cracks from 1 to 6 in. wide, having the appearance of tension cracks, because there is no depression, and are some distance (from 100 to 400 ft.) in advance of the actual subsidence. When the approaching subsidence from a former crack is not far distant, another tension crack develops farther to the west and the cycle is repeated. In explanation of these occurrences the following theory is offered:

The dome of caving, working upward from the mining operations advancing to the west, undercuts an unsubsided portion of the surface, producing in the rock a cantilever or projecting beam unsupported at the eastern end. As the dome moves upward the thickness of the cantilever is decreased, and at some indefinite point the weight of the overhanging portion overcomes the cohesion of the rock and the end of the beam falls, possibly until supported by the broken material within the dome and the subsided material pushing in from the east. The vertical drop is evidently small and a tension is produced at the ledge or upper edge of the beam, resulting in the aforementioned tension crack at surface. The broken cantilever then fails in a series of parallel shear planes, thus producing the terraced effect on surface. The outside crack at surface remains undisturbed until the westward advance of mining operations produces a dome of sufficient height to repeat the operation.

The similarity of conditions (except for the difference in depth) at the properties would seem to indicate that the Athens subsidence should advance from west to east in like manner. East of the Athens cave and above the workings in blocks 2 and 3 is produced the first cantilever, and apparently a tension crack may be expected to the east of the present cave as the first indication of future subsidence. It will be interesting to learn whether the Athens cave follows these mechanics of subsidence, and how successful will be the attempt to predict the nature of future subsidence by comparison.

Among the statistical features of the Athens cave are some that are rather unusual. As mentioned before, the depth of opening, in relation to its depth below surface, was a little less than 1 in 10. From this it would seem that the critical depth, at which underground operations will not affect the surface, is a variable, largely dependent upon geological conditions. The slope of the lines, in transverse section, drawn from the limits of excavation to the limits of cave at surface, is at an angle of 91° on the north and 95° on the south. This necessitates a widening of the statements of some writers who say that there is always a draw, the angle of which varies from 60° to 85°, depending on the nature of the overlying rocks and the inclination of the strata. The cross-sectional area at the deepest part of the surface cave is about one-third that of the mined section below.

As stated, the purposes of the Athens system of mining were to delay a surface cave and to locate the eventual cave over a mined area below the working places. Because of the depth of mining, the present cave was unexpected, nevertheless the conditions described above have been nearly fulfilled. Even though the limits of the cave are partly over the uncompleted block 2, the total result to date has been merely to add more water to a portion of the mine that has been prepared and operated as a wet area. It is unfortunate, from the pumping standpoint, that the cave occurred as early as it did, but fortunate from the mining standpoint, because of its results. A large proportion of the ore in block 2 has already been mined, and if the present high percentage of dry working places can be made to persist in spite of the cave, conditions will be better than in block 1; and if the theoretical cantilever capping over block 3 and the future blocks 4 and 5 and 6 shall fail only when mining is nearing its end within the block, the Athens system of mining will be truly successful.

DISCUSSION

(*George S. Rice-presiding*)

W. R. Crane,* Washington, D. C. (written discussion).—In 1928, I made an extended investigation of rock structure in the iron mines of Ishpeming and Negaunee, spending eight weeks on the surface and underground. This work included the Maas-Negaunee, Athens and Cambria mines at Negaunee, and the Holmes, Section 16 and Barnum mines at Ishpeming. The results of the investigations were given to the respective companies and later published in *Bulletin* 295 of the U. S. Bureau of Mines.

The angles of draw were found to be 80° and 85° at Ishpeming and Negaunee, respectively, both angles being the ultimate angles while the initial breaks were practically vertical. These angles also include a factor of safety. The angles of draw in glacial drift were found to be 35° and 45°, the former in sand and gravel, the latter with sand, gravel and clay. The combined angles of draw in rock and drift indicate the extent of surface disturbance resulting from subsidence.

It is interesting to note the close accord reached by two investigators in the determination of draw in the same locality, which probably is due to the employment of "existing geological conditions" in the studies made.

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* U. S. Bureau of Mines.

Yieldable Metal Props for Underground Support

By Roland D. Parks,* Houghton, Mich.

(New York Meeting, February, 1933)

To construct a yieldable metal prop of demonstrated practicability has been the aim of the writer of this article for a period of years. Such a prop is herewith described; it involves a yielding principle not previously utilized, to the best of the author's knowledge, in any of the numerous types of props which are in use in Europe and England or on record at the United States Patent Office.[1]

No attempt will be made in this paper to analyze the situation in regard to the economics of timber substitutes. Suffice it to say that shortage of suitable timber combined with high freight rates has already presented a problem in some mining districts, and in the relatively near future will probably demand considerable attention from the mining industry as a whole.

In order to understand the problems involved in the design of timber substitutes for underground support, it will be well to review briefly the fundamentals of ground subsidence and of the action of timber as props.

The forces encountered in the support of mine openings are not computable. As a rule subsiding ground moves slowly; but it moves with an irresistible force. The only way to be certain of stopping ground movement is to completely fill up the openings below such movement. This is, obviously, an impracticability from the mining standpoint. The ordinary procedure is to put in timber to hold up the loose back and permit the natural arch above to resist its own loading. Under these conditions the timber rigidly supports the loose; but if the natural arch fails, the timber is then called upon to resist this additional pressure. Usually it cannot do this. The result is a crushing of the supports. Efficient timbering is that which will last just long enough to complete in safety the mining operations necessary to extract the ore.

Wood has a cellular structure. It has great longitudinal strength for its weight. Further, it possesses the ability to maintain a high percentage of its strength while it is being crushed. In other words, it rigidly supports loads up to its capabilities and then continues to sustain from 50 to 80 per cent of this loading while it yields and shortens. In this respect wood is at distinct variance with other common structural materials. It shows the effects of subsiding ground by bending, cracking, brooming, and splitting while it still supports a considerable load. It is this property together with the ease with which it is framed that makes wood so desirable for underground support. Wooden props do not ordinarily snap and break suddenly.

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* Associate Professor of Mining Engineering, Michigan College of Mining and Technology.

[1] U. S. Patent 1677796 issued to the author, July 17, 1928.

The object of the author's work on timber substitutes has been to create in a metal support this yielding action of wood. To accomplish this objective the support must stand as a rigid column up to a definite loading and then yield; it must be so designed structurally that it will fail by yielding rather than by column action. In addition the yield point should be uniform throughout the life of the support, or, in other words, at all points within the deformation limit of the column. Mr. George S. Rice,[2] in a recent article on ground movement in coal mining, said, "It is manifest that the ideal steel prop is one that will slowly but regularly yield with uniform resistance." Also, the support must be practical as to cost, installation (ease of placement, ease of replacement), and life. It should be simple, rugged, and fireproof, and made of ordinary materials; further, it should be capable of fabrication in a mine shop.

Substitutes for Timber

A brief summary of the work of others on this problem shows that various substitute props are already either in use or proposed for use. They may all be grouped into two classes: rigid and yieldable.

Rigid metal props, constructed, as a rule, from standard structural shapes, have only a limited field of use in ground where there is little or no movement but where there is a tendency for some loose to develop. They cannot be used in subsiding ground, because they will either fail suddenly, or, if they are exceedingly strong, will break the subsiding strata and cause bad falls of ground. In any case the failure of rigid metal props will cause a tangled mess, which will probably not be recoverable except by the use of an acetylene torch—an item of additional expense and, in some mines, an unsafe practice.

Yieldable props may be subdivided into three groups as follows:

1. *Non-telescoping yieldable props.* The best example of this type is the iron pipe prop having wooden plugs in the ends. These plugs broom under excessive loading and thus provide the yield feature.

2. *Telescoping resistance props.* Props of this type are built up of two interslidable elements with a third element offering resistance to the telescopic action. Peat fiber, granular material, wood, and crushed stone are the most commonly used yieldable elements.

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[2] G. S. Rice: Recent Research on Ground Movement in Coal Mining. *Trans.* A. I. M. E. (1932) **101**, 280.

3. *Telescoping friction props.* Two-element intersliding props of this type, depending entirely upon friction for resistance to intersliding action, are suitable for low to moderate loadings only.

The author chose to conduct his experiments along the line of group 2 above, for the following reasons: (*a*) Props of the type included in group 1 are not entirely substitutes, since wood plugs are used at the ends of the intermediate pipe to provide yield; (*b*) props of group 3 (frictional props) are not reliably uniform in their resistance action, and may slip at lower loadings than estimated. They may also become rusty, and their resistance may mount to the point where the prop will fail by column action rather than by slippage.

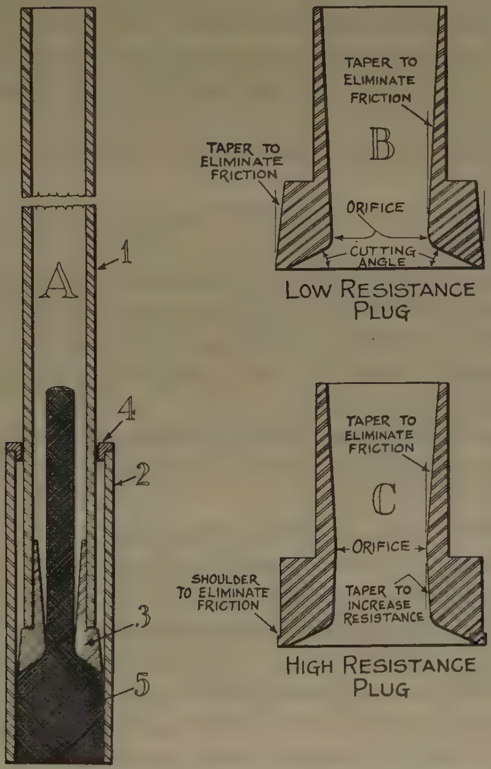
A
1
4
2
3
5
TAPER TO ELIMINATE FRICTION
B
TAPER TO ELIMINATE FRICTION
ORIFICE
CUTTING ANGLE
LOW RESISTANCE PLUG
TAPER TO ELIMINATE FRICTION
C
ORIFICE
SHOULDER TO ELIMINATE FRICTION
TAPER TO INCREASE RESISTANCE
HIGH RESISTANCE PLUG

Fig. 1.—Section views of yieldable metal prop assembly and details.
1 and 2, interslidable iron or steel pipes.
3, plug fitted to bottom of inside pipe.
4, guide collar to keep pipes in alignment.
5, yieldable resisting metal.

The Parks Prop

The prop which the author has developed consists, in principle, of two intersliding elements whose telescopic action is resisted by a third element. Metal or metallic alloy is used throughout. The intersliding elements are each rigid; the resisting element is yieldable. The yield point of the third element is, at all times, below the column strength of the support as a whole; that is, the column strength of the interslidable structure.

Construction is extremely simple, as shown in cross-section in Fig. 1*A*. Parts 1 and 2 are interslidable iron or steel pipes; part 5 is the yieldable resisting metal which is forced by longitudinal pressure through an orifice in a plug (part 3) loosely fitted to the bottom of the inside pipe; part 4 is a guide collar between the two pipes to keep them in alignment.

For laboratory purposes a model prop was constructed of the following pieces:

Part 1. Seamless steel tube 16 in. long, 2 in. outside by 1½ in. inside diameter.

Part 2. Standard extra-strong 2½-in. steel pipe 15½ in. long.

Part 3. Plug of ordinary or high-carbon steel not hardened, the orifice diameter, cutting angle, and tapers varying in each test.

Part 4. Cast-iron bushing. Not necessary if interslidable pipes fit closely one within the other.

Part 5. Lead such as was available at the time, of which the approximate analysis was: Pb, 98.0 per cent; Bi, 0.80; Sn, 0.65; Zn, trace; Sb, trace.

Clearance of about 1/100 in. was allowed all around.

It will be obvious at the completion of this description that, with no change in principle of action, other standard or special structural shapes could be substituted for the pipes used, and other metal or metallic alloy for the lead as a resisting element. The outward appearance of the prop is a minor matter and can be varied to suit the structural conditions necessary for its installation. The yield principle is the important feature.

In this prop metal is stressed to failure in the resisting element to permit the unstrained telescoping elements to yield and thus "give" under excessive loading. Metals fail by shear or flow. Some metals fail by a combination of these two characteristics; others by only one.

Plastic or viscous materials are those which will flow without showing any other indication of failure. Brittle or comminuible materials are those which will crush to a powder, or crumble to pieces, or fail by shearing on definite angles under a compressive load. In the former class are such materials as wrought iron, soft and medium steel, the alloys, lead, copper, zinc, and the like. Of the latter class are cast iron, hard or tempered steel, brick, stone, cement, etc.[3]

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[3] Johnson: Materials of Construction. Rewritten by M. O. Withey and James Aston. Ed. 7, 14. New York, 1930. John Wiley & Sons.

Thus in the design of a prop of this type, wherein strained metal provides the yield, this metal must be of the plastic or viscous class, if the prop is to yield at a uniform loading.

Brittle metals, even though their breakdown points may be uniform, cannot give a uniform yield, because the granulating effect, after failure, continuously changes the sizes of the particles. This point is of particular importance, for many of the telescoping metal props already in use depend upon granulated material such as sand for their yield.

Regulation of the yield point in the prop is accomplished through the proper coordination of two mediums, as follows:

1. The design of the plug (part 3) may be varied within structural limits in regard to orifice diameter and inside and outside tapers.
2. The hardness of the plastic resisting element (part 5) may be varied by a change in metal composition.

Fig. 1*B* shows in detail the type of plug that proved most satisfactory in the laboratory tests of the model prop. Medium to high-carbon steel is used for this piece. The plug is tapered both inside and outside, because these tapers are necessary in order to reduce frictional resistance to a minimum. They need not be of any definite amount so long as they are in the proper direction. Friction, as such, is extremely difficult to control. It produces wide, unexpected fluctuations in the yield point of the prop, and therefore must be eliminated to permit the prop to act in a manner that is capable of calculation within the limits of structural design.

The high-resistance plug shown in Fig. 1*C* has a slight taper converging to the orifice in order to increase the work by keeping the yieldable metal in a state of flowage for a longer period of time. Friction is eliminated by tapers, as mentioned above, although the same effect may be produced by a recessional shoulder as shown on the sketch. A shoulder of this type could be used on the inside of the plug immediately above the orifice, but is not shown because on a lathe it is easier to machine the inside as a taper.

While the high-resistance plug has not been fully tested in the experimental work to date, it has demonstrated that the most sensitive point at which to increase resistance, and so raise the yield point, is immediately below the orifice.

The cutting angle, or angle of incidence, of the lower face of the plug (Fig. 1*B*) need not be of any particular amount, although from the tests so far conducted it is apparent that it is better if flatter than 45° to the horizontal. Cutting angles of 0°, 12°, 23°, 35° and 45° have all been tried in the same test piece with practically no change in the stress-deformation curve. The 45° cutting angle places undue stress upon the walls of the plug, and the 0° angle creates more side pressure on the lower pipe (part 2); hence it is concluded that the best angle of incidence lies somewhere between these two extremes. In the design of heavier props this deduction might be subject to revision.

The stress-deformation curves of Fig. 2 show the action of the yieldable metal prop under discussion in comparison with a white pine prop and with the theoretically ideal support. For the metal prop (curve *A*) the load mounts rapidly, with only slight deformation, to the full resistance of the yielding element, drops about 10 per cent when the plastic metal begins to flow through the orifice, and then continues to yield at a uniform resistance throughout the entire limit of the design. As shown by curve *C*, wood loses in ultimate strength each time it is stressed beyond its elastic limit, with the result that its resistance drops off rapidly when subjected to successive or continuous yielding. The ideal stress-deformation curve, *B*, rises vertically to the limit of resistance, and then yields in a perfectly straight line.

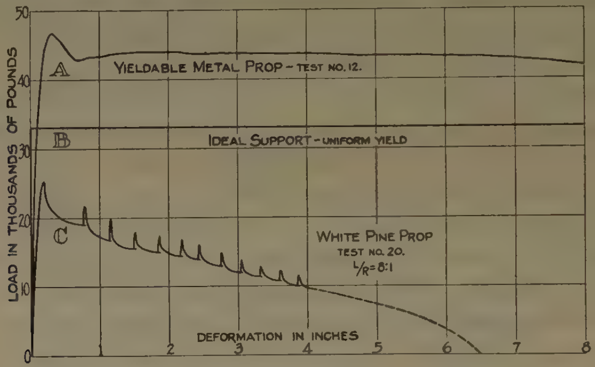

FIG. 2.—STRESS-DEFORMATION DIAGRAMS.

A comparison of the diagrams shows that the metal prop does not vary appreciably from the ideal, whereas wood shows itself to be far from perfect. After flowage starts, the maximum deviation of the metal prop curve from the ideal mean is about 2 per cent, a variation that is well within practical limits. The particular curve shown for the metal prop is the result of one of several laboratory tests, all of which produced curves so nearly identical that it was deemed advisable to show only one as representative of the group.

At this point it may be well to explain how curve *C* was obtained. An ordinary laboratory testing machine of the Riehle type was used for all runs. With wood posts, it was found that a continuous run produced misleading results, because the wooden post simply resisted pressure up to its maximum strength with only slight deformation, and then collapsed as quickly to zero. Consequently, to test timber for the type of loading it receives when used to support heavy ground, the closest approximation is obtained by loading the prop to its yield point, cutting off the load movement to allow the prop to adjust itself to static pressure, and then successively repeating this cycle until the prop has failed completely.

CONCLUSIONS

From the results already obtained, the author feels justified in drawing the following conclusions:

1. Yield action of sufficient uniformity for all practical purposes is obtainable by use of the principle outlined; that is, by utilizing the strain of viscous metal to obtain the yield.

2. Sufficient regulation is available to permit the design to be adapted to all ordinary conditions of underground support.

In addition to the general conclusions mentioned above, the tests have shown that the prop is practical from a construction standpoint for the following reasons:

1. The design may be constructed of standard materials without recourse to special shapes except for extraordinary conditions of framing.

2. Any good machinist can construct the prop, and any good timberman can install it. No special refinements are necessary for successful operation.

3. Original cost, compared with timber at present prices, is relatively high; but replacement cost is extremely low, requiring only the recasting of the yieldable metal. Nothing is destroyed or lost in the yielding operation.

4. The interslidable elements, barring accidents, can be used indefinitely, or until they rust out.

5. Size of top pipe, bottom pipe and plug can be readily standardized at a mine to suit local conditions.

6. The prop is fireproof.

Cost of construction is estimated to vary from about four dollars for a small prop to about fourteen dollars for a large one. This estimate is based upon carload prices for extra-heavy pipe delivered at Houghton, Mich., and assumes that fabrication will be done in the average mine shop. An itemization of these figures is approximately as follows:

1. Small prop, 5 ft. high, yield point about 30,000 lb., using a 2½-in. top pipe and a 3-in. bottom pipe cast 12 in. deep with resisting metal:

Top pipe 2½ in. by 4 ft. @ 33¢ per foot	$1.32
Plug, including labor of machining	0.75
Bottom pipe 3 in. by 2 ft. @ 44¢ per foot	0.88
Resisting lead alloy, 25 lb. @ 0.03	0.75
Casting of metal—labor	0.25
	$3.95

2. Large prop, 8 ft. high, yield point approximately 100,000 lb., using a 5-in. top pipe and a 6-in. bottom pipe cast 12 in. deep with resisting metal:

Top pipe 5 in. by 7 ft. @ 90¢ per foot	$ 6.30
Plug, including labor of machining	1.00
Bottom pipe 6 in. by 2 ft. @ $1.23 per foot	2.46
Resisting metal, 125 lb. @ 0.03	3.75
Casting of metal—labor	0.25
	$13.76

In each of these estimates the labor charge is approximately one dollar. If a large number of props are constructed at one time, this figure can be reduced somewhat by standardization of material, and by having the plug forms stamped roughly to size, so as to reduce the machine work necessary to finish them.

The prop lends itself admirably to standardization, when one considers the possibility of grouping several small to medium-sized props into a unit, wherever it is necessary to install a heavy support. Except in extraordinary cases, these groups would consist of three, four, five, or six props, all of the same size, fastened around a central standard to keep them in alignment. The central member need consist only of a pipe about 2 ft. shorter than the top prop pipe and having crossplates welded to it at about 2-ft. intervals. These crossplates would be cut triangular, square, pentagonal or hexagonal in shape and would have a central hole to fit around the central standard and semicircular peripheral indentations aligned one above the other to receive the props. Binding straps would be placed around the structure at each crossplate.

The effect of such a grouping of props would be to cut down the effective column ratio and thus lend strength to the interslidable elements of the props by eliminating column action. Convenient sizes of props for such practice would be as follows:

1. Top pipe 2½-in. extra strong fitting inside a 3-in. extra strong bottom pipe. The top pipe can vary in length from 3 ft. for a 4-ft. prop to 7 ft. for an 8-ft. prop. The bottom pipe always is 2 ft. long. No bushing is required with these pipe sizes.

2. Top pipe 3-in. extra strong, fitting, by means of a bushing, within a 4-in. bottom pipe. The lengths given above apply here.

Another easy method of grouping props would be to have interlocking lugs fastened to each prop at certain intervals. This would be simpler than the central standard for some installations and just as effective as regards reduction of column ratio.

In addition to the properties mentioned of the support under investigation several other conclusions have been drawn in relation to this particular prop and to other forms of it, embodying the same basic principle of action:

1. The cutting angle of the plug, at least with lead as the resisting metal, apparently makes little difference.

2. Mild steel in shear serves as a good resisting element for high loadings but, from results obtained, apparently is too erratic for props called upon to resist less than 50 tons. Two props utilizing steel in shear gave excellent deformation curves for a yield of approximately 4 in. at 115 and 160 tons respectively. These props, however, were complicated to build, requiring a high degree of accuracy in fabrication, and therefore were dropped temporarily from immediate investigation. They would be suitable only for special installations at shaft plats and along main drifts.

3. Frictional resistance must be eliminated from the prop, because it is an indeterminate quantity and makes for irregular and unexpected fluctuations in the yield point. This was quickly recognized in the course of the experiments, and the plug was tapered to counteract its effect. Resistance to flowage was later put back to a point at which it could be controlled by converging at a flat angle the entrance to the orifice of the plug.

The experimental work outlined here is far from complete. It has progressed only to the point where it has proved to be of practical application. Much more study will be necessary in order accurately to design props to meet specific conditions. Future work will include tests of larger and multiple props, of other metals and alloys as the yielding element, and of varying plugs and orifice diameters. The use of standard rolled sections instead of pipe and the design of framing connections will also be studied.

Effect of Approximately Vertical Cracks on the Behavior of Horizontally Lying Roof Strata*

By P. B. Bucky,† New York, N. Y.

(New York Meeting, February, 1933)

In previous publications[1] it was shown that a scalar model of any weighty structure, where the stresses produced are mainly due to gravitational forces, will behave similarly to its prototype if the model and prototype material are the same and the model is placed in a centrifugal field of force (substituted for the gravitational field) which has been increased in the same proportion as the linear model scale is decreased. This principle is now being applied to a study of the behavior of mine structures in an attempt to determine the laws that control them.

The importance of using models as a means of solving mining problems has been discussed previously, yet it may be well to reiterate here that this method of research and type of solution have been productive of far-reaching results in other engineering fields, notably those of hydraulics and aerodynamics.

Model research is a comparatively cheap and time-saving method of checking practical experience. Up to the present time the best guide to what will happen underground has been the judgment of the man who has observed most and been longest acquainted with the local conditions. His experimental work underground is limited by the elements of cost, time and danger, while failure is seldom permissible. In a model, as in a test tube, all elements that affect the solution of a problem are under definite control. Each element may be varied independently at will and its effects noted. Experimental work on a small-scale model may therefore be performed at a comparatively small expenditure of time and money, instead of on a full scale in the mine, where the cost, time, and danger elements are necessarily great. The experiments described in this paper are to be considered as part of a progress report and a continuation of *Technical Paper* 425. One object of the research program is to make each experiment point the way for further research on that particular phase of the general problem. The results and conclusions presented here are therefore not to be considered as final, but are to be interpreted as

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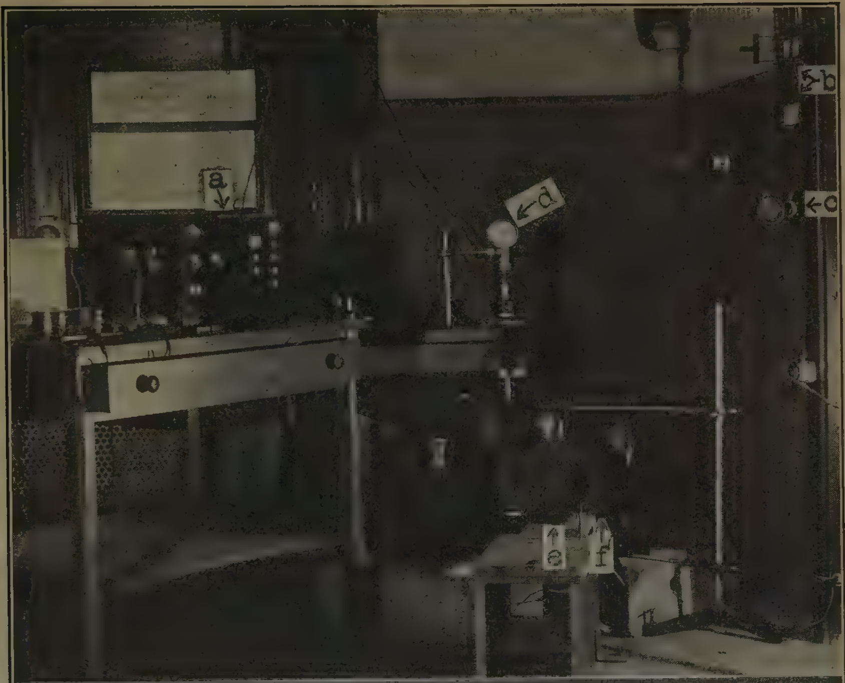
* A research project of Engineering Foundation and Columbia University.

† School of Mines, Columbia University.

[1] P. B. Bucky: The Use of Models for the Study of Mining Problems. A. I. M. E. *Tech. Paper* 425 (1931).

The Use of Models under Centrifugal Force. *Trans.* Amer. Soc. Mech. Engrs.



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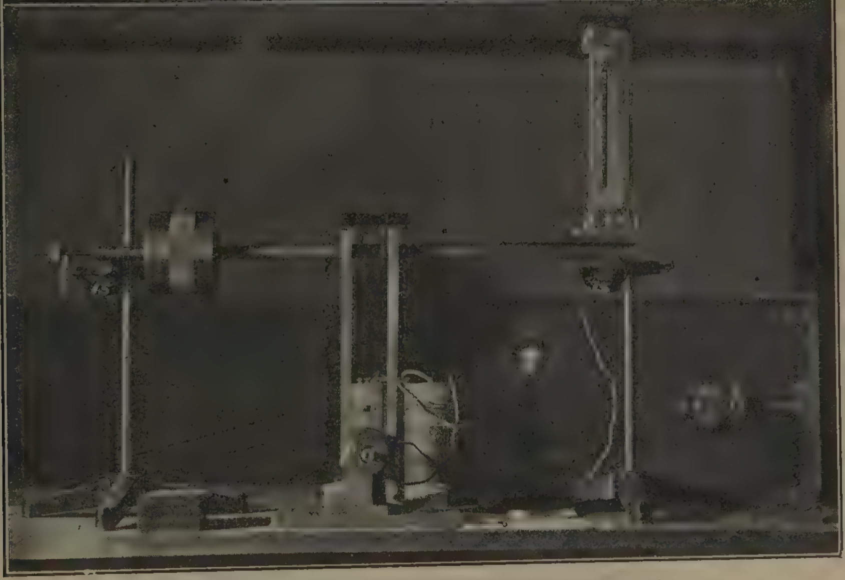

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FIG. 1.—CENTRIFUGE AND STROBOSCOPIC LIGHT.

a. Light-control circuit.	*d.* Tachometer.
b. Motor switches.	*e.* Camera.
c. Speed control.	*f.* Mercury tube.

FIG. 2.—BALANCE.

applying only to the material and type of structure specifically mentioned. Although reasoning from the results of experiment may develop some tentative conclusions, these will be marked as such, and probably will require more experimentation to obtain conclusive evidence.

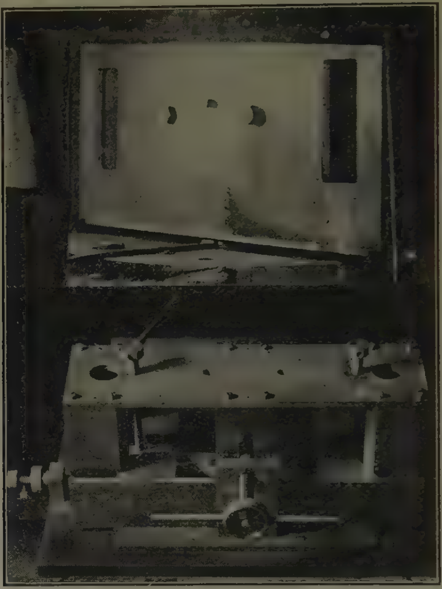

Fig. 3.—Rotating box with observation ports.
Fig. 4.—Pouring and centering device.

Equipment

The present equipment (Figs. 1 to 4) is essentially the same as previously described. A stroboscopic[2] mercury arc light has been applied to simplify the problems of observation and photography, a moving picture camera being arranged to run at correct speeds for photographic purposes. A balancing device makes it possible to run a test with one instead of two models; rock-cutting and grinding equipment, and a screen and projection apparatus for making deflection measurements have also been designed and the parts either built or purchased. Some work is yet to be done in developing accessory equipment and it is the intent, when it is more nearly complete, to publish in detail all information concerning equipment and design and test procedure.

Experimental Procedure

The procedure followed in this experiment was first to obtain the effect of a vertical center roof crack, with sides a definite distance apart. This

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H. E. Edgerton: Stroboscopic Moving Pictures. *Elec. Eng.* (May, 1931).

distance was then decreased until the sides of the crack were in juxtaposition. The roof was then cracked in a number of places and the results noted. The effect of filling the cracks with grout, where they are not in juxtaposition, was then obtained. The idea of grouting roof cracks appears to be of much importance in mining and tunnel work, as the results of tests show that it increases the roof strength to and beyond that of a solid roof. To check the behavior of these models, the data on the model behavior of a solid roof were obtained. While all photographic

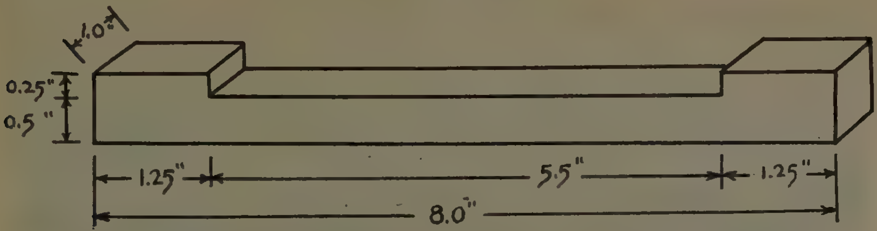


Fig. 5.—Standard limestone base used for models in this series.

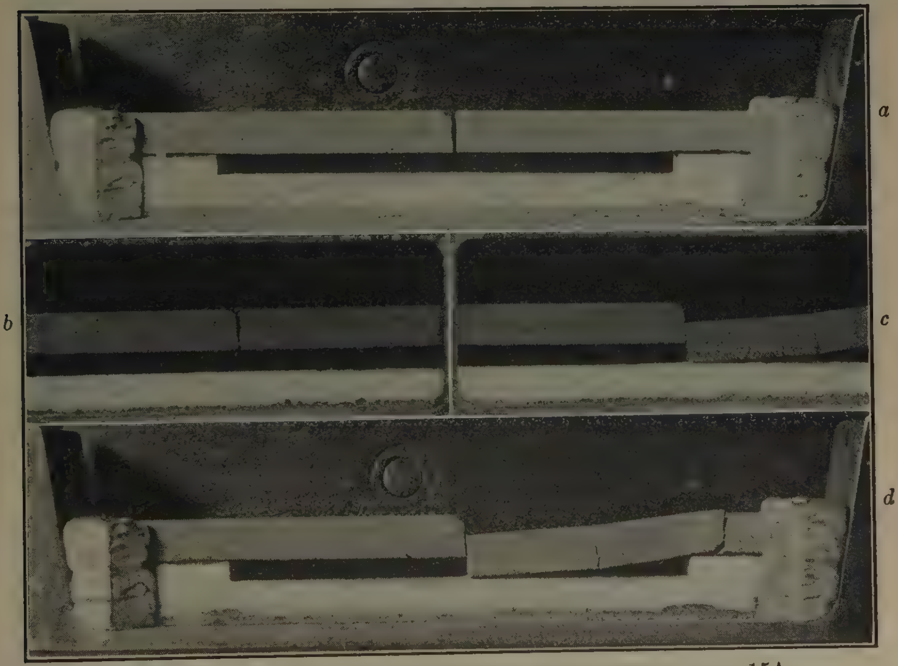


Fig. 6.—Photographic results of roof-crack tests, model 15A.

a. Before placing in centrifuge.
b. In centrifuge, while running; 400 r.p.m., 36.3 model ratio.
c. In centrifuge, while running; 700 r.p.m., 111 model ratio.
d. After breaking.

data are not presented in this paper, they are available for inspection in the mining laboratories at Columbia University.

Preparation of Models

To study this subject seven models were prepared as follows: the limestone base was the same for all, and of dimensions and shape as shown in Fig. 5. For model 15A (Fig. 6) the cover consisted of two pieces of sandstone ½ in. thick and 1 in. wide, of a total length of 8 in., with a 0.05-in. opening between them at the center. Model 15B (Fig. 7) was the same as 15A, except that the opening width was 0.008 in. Model 15C (Fig. 8) was the same as 15A and 15B, except that the ends at the center abutted, each end being lampblacked to give as close-fitting and frictionless a joint as possible. For model 15D (Fig. 9) the cover was a solid piece of sandstone, 8 by ½ by 1 in. For models 15E and 15F the cover was cracked in various places as shown in Figs. 10 and 11. The method of assembling 15E, with five cracks, one of which was at the center, was as follows: The left endpiece of the cover was attached to the base with library paste, the model was then centered in the holder and type

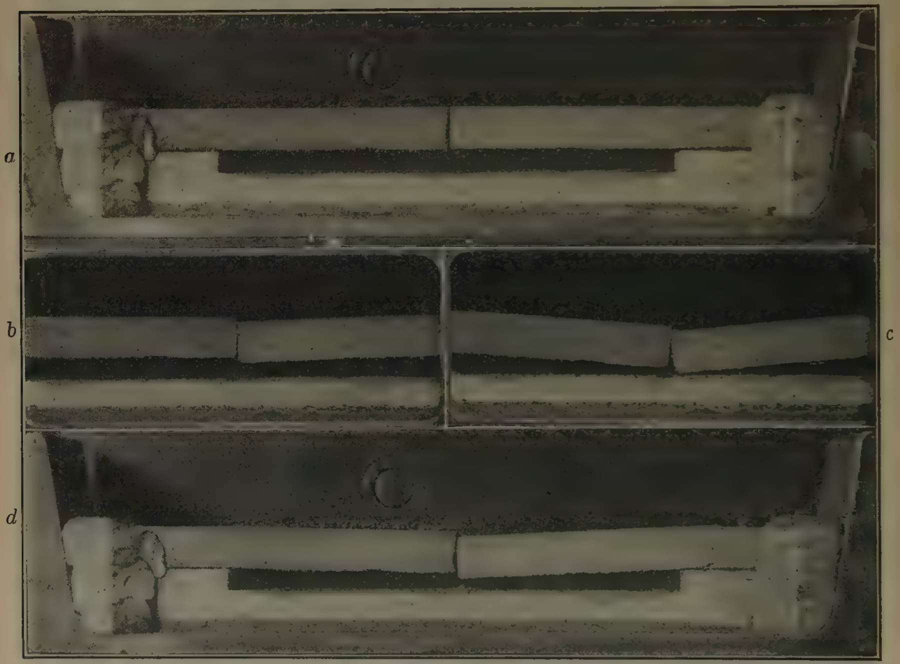


Fig. 7.—Photographic results of roof-crack tests, model 15B.
a. Before placing in centrifuge.
b. In centrifuge, while running; 700 r.p.m., 111 model ratio.
c. In centrifuge, while running; 1200 r.p.m., 326 model ratio.
d. After breaking.

metal poured around the left end. The remainder of the cover was put into place and held there by means of a screw through the right-hand end of the holder. Type metal was poured in this end of the holder and the screw removed. No adhesive was used in the cracks. The cover of model 15F was cracked in only four places, there being no crack at the center. It was assembled in the same manner as 15E.

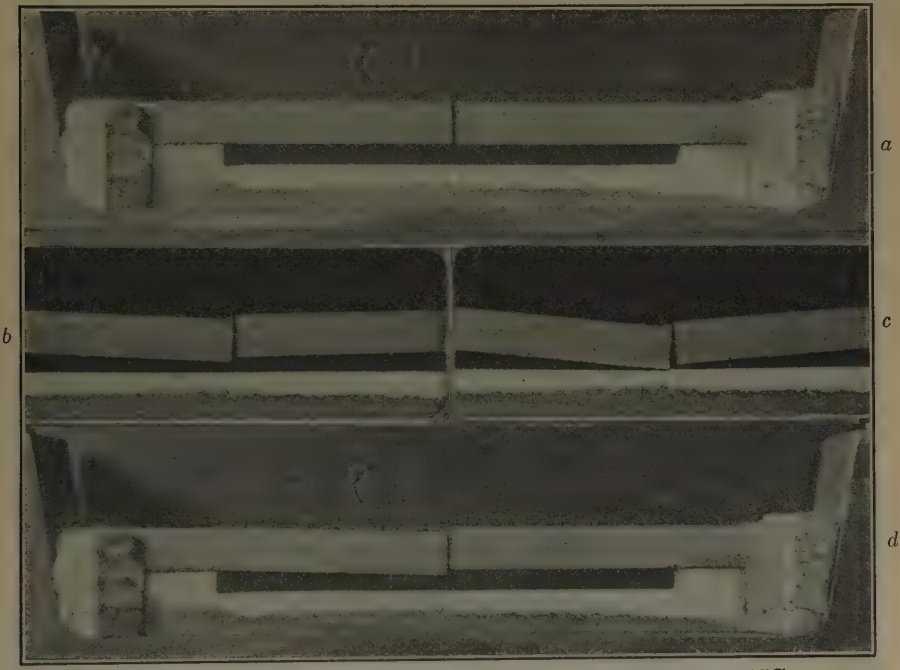


Fig. 8.—Photographic results of roof-crack tests, model 15C.
a. Before placing in centrifuge.
b. In centrifuge, while running; 1200 r.p.m., 326 model ratio.
c. In centrifuge, while running; 1500 r.p.m., 310 model ratio.
d. After breaking.

Model 15G (Fig. 12) was built exactly similar to model 15A, but in the opening neat cement was placed for studying the effect on roof behavior of grouting roof cracks. If the effect of open roof cracks can be minimized by using cement grout or a filling of some other substance, an important means of attacking underground problems of this type is available.

Tests and Measurements

The models were run in the centrifuge at speeds shown on the data sheet, Table 1. At these speeds photographs were taken. The camera was in the same position and focused in the same way for all exposures. The image on the photographic plate was then thrown on a ruled wall screen by means of a projection lantern, the plate taken at 0 r.p.m. serving as a reference plate for each set. The method of making readings was as follows: The plate for the 0 r.p.m. reading was first flashed on the screen and the amount of enlargement adjusted. Lines were then carefully penciled on the screen, of cross-section paper, to show the surface, mine roof, bottom, and the model center in a vertical direction.

The plate taken at 400 r.p.m. was then flashed on the screen and centered left and right, care being taken to make the roof, bottom, and surface lines noncoincident. For a reading at the center, if it is assumed that the bottom is 0.5 in. below that of reference plate and the surface is 0.58 below that of the surface on the reference plate, the surface at the center has therefore subsided 0.08. In this manner readings were taken while rotating of actual deflections to 0.001 in. or less, depending on the amount of enlargement and the ruling of the screen. In Table 1 subsidence readings for models 15A, B and C were made at the immediate right and left of the center. For models D, E, F, and G readings were made at the center. Any other readings may be made by using again the same plates.

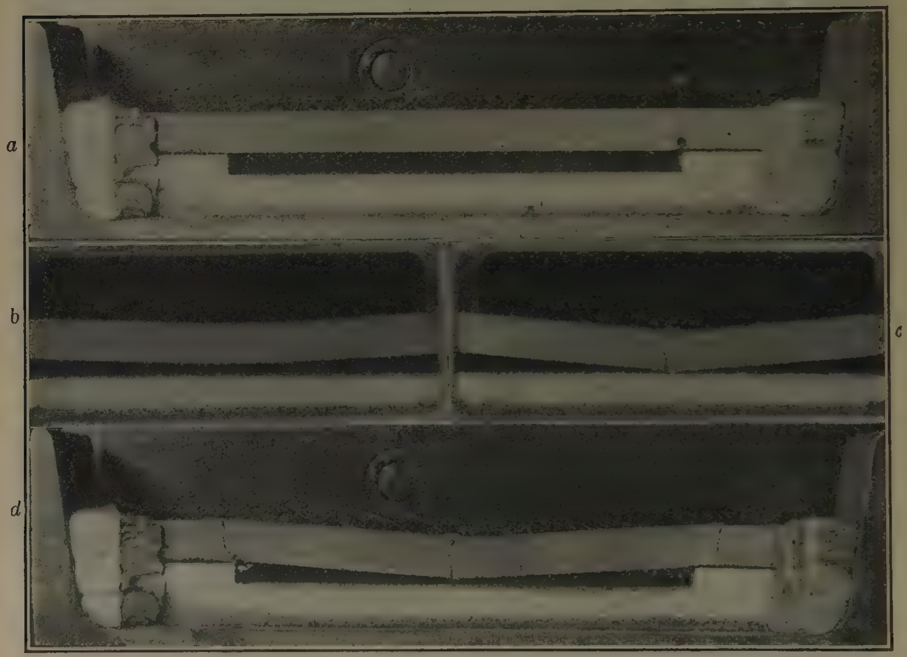


FIG. 9.—PHOTOGRAPHIC RESULTS OF ROOF-CRACK TESTS, MODEL 15D.

a. Before placing in centrifuge.
b. In centrifuge, while running; 1500 r.p.m., 510 model ratio.
c. In centrifuge, while running; 1600 r.p.m., 580 model ratio.
d. After breaking.

The results in Table 1 are those obtained in the laboratory. The actual field results would be obtained by multiplying the laboratory results by the model ratio. The relationship of revolutions per minute to model ratio is shown in Table 1. For example, in model 15A at 400 r.p.m., or a model ratio of 36.3, the center subsidence was 0.006 in. The field subsidence would be 36.3 × 0.006 in. = 0.218 in. Model 15E at 1400 r.p.m., or a model ratio of 444, subsided 0.112 in. The field subsidence would be 444 × 0.112 = 49.73 in. or 4.14 ft.

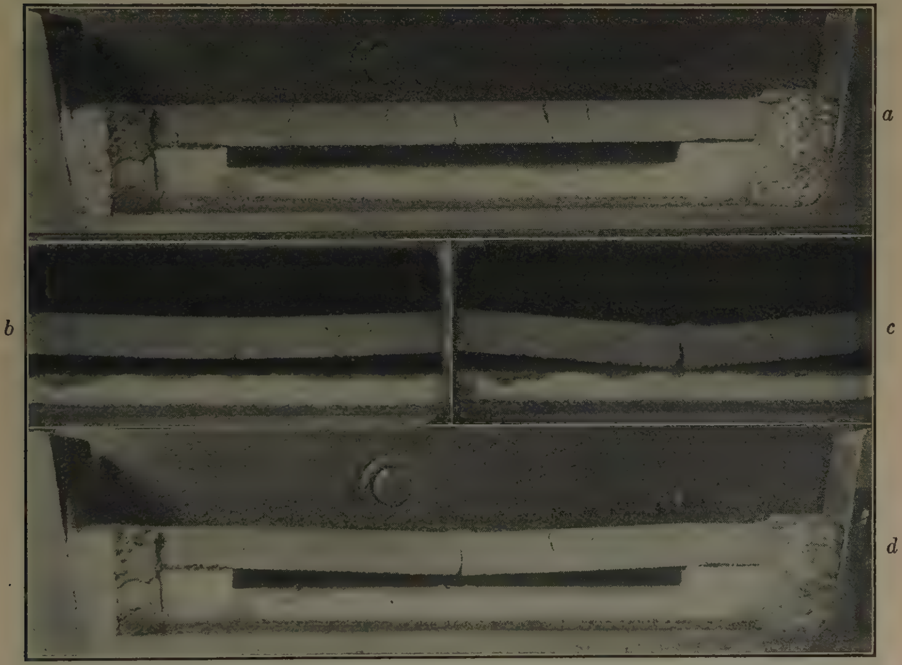


FIG. 10.—PHOTOGRAPHIC RESULTS OF ROOF-CRACK TESTS, MODEL 15E.
a. Before placing in centrifuge.
b. In centrifuge, while running; 1200 r.p.m., 326 model ratio.
c. In centrifuge, while running; 1500 r.p.m., 510 model ratio.
d. After breaking.

In comparing the model behavior, the standard of comparison is the behavior of 15D, the one with the solid cover. This was run at 400, 700, 900, 1200, 1400, 1500 and 1600 r.p.m., corresponding to model ratios of 36.3, 111, 183, 326, 444, 510 and 580, at which it broke. In Figs. 6 to 9, it may be observed: that 15A, with a 0.05-in. center crack, broke at 700 r.p.m., or at a model ratio of 111; that 15B, with a 0.008 crack, broke at 1200 r.p.m., or a model ratio of 326; and that 15C, with a well lubricated crack of no width, broke at 1500 r.p.m., or a model ratio of 326. The same figures show that as the ends of the roof begin to bear against each other there is a tendency to support one another and to transmit the pressure to the vertical uprights of the holder. Model 15E, whose roof had a series of five cracks, one of which was at the center, failed at 1500 r.p.m., or at a model ratio of 510. In model 15F, four cracks were present, but none at the center, and this model failed at 1500 r.p.m., or a model ratio of 510.

Effects of Crack Width and End Fixity

By comparing the behavior of models 15A, 15B and 15C, by means of the photographs Figs. 6 to 9, and from the data in Table 1, it is seen that the smaller the crack width and the greater the opportunity of the two cantilever beams to act on each other the stronger the roof structure. Model 15A, with a 0.05-in. opening, failed at 700 r.p.m. Model 15B, with a 0.008-in. opening, failed at 1200 r.p.m., and model 15C, with a 0.000-in. opening, failed at 1500 r.p.m. As the two sides of a crack deflect and act on each other in these models, pressure is exerted along the roof toward the model ends. This end thrust exerted on the strata under stress is therefore of decided importance. If there is little or no end thrust, the roof will fail with a much shorter span.

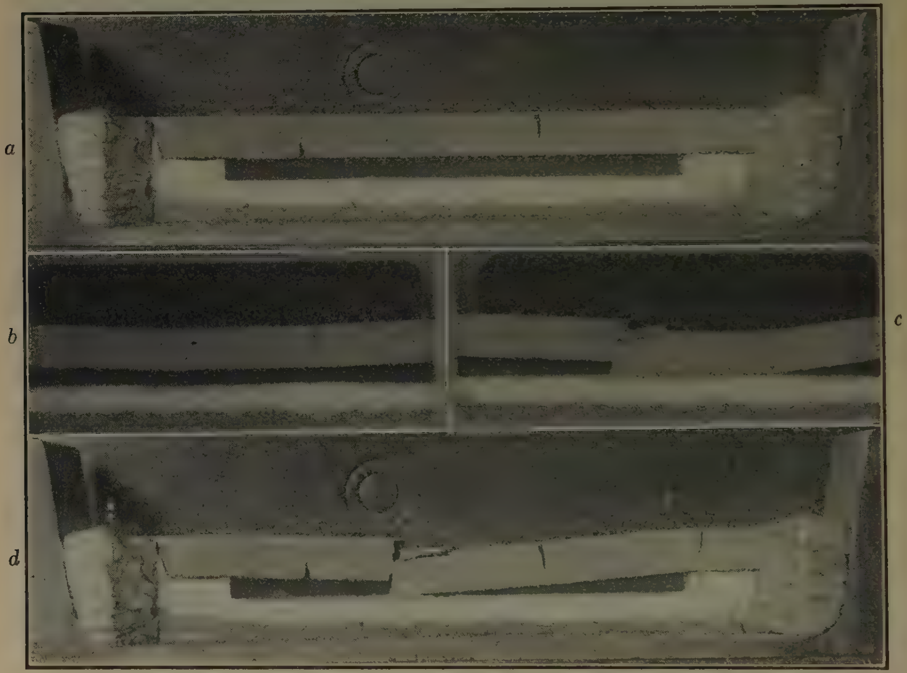


Fig. 11.—Photographic results of roof-crack tests, model 15F.
a. Before placing in centrifuge.
b. In centrifuge, while running; 1200 r.p.m., 326 model ratio.
c. In centrifuge, while running; 1500 r.p.m., 510 model ratio.
d. After breaking.

TABLE 1.—*Data Sheet*[a]

Speed,[b] R.p.m.	Model[b] Ratio	Subsidence; In.							
		Place of Readings[c]	15A	15B	15C	15D	15E	15F	15G
0	0	L	0	0	0	0	0		
		R	0	0	0	0	0		
400	36	L	0.006	0.004	0.006	0.003	0.006		
		R	0.006	0.006	0.006			0.0125	0.0031
700	111	L	0.025	0.038	0.025				
		C				0.006	0.0125	0.0187	0.0125
		R	0.250	0.100	0.019				
900	183	L		0.069	0.106				
		C				0.031	0.038	0.0438	0.0312
		R		0.138	0.038				
1200	326	L		0.192	0.132				
		C				0.050	0.075	0.0875	0.0562
		R		0.250	0.063				
1400	444	C		No pictures taken			0.112	0.1125	0.0750
1500	510	L			0.250				
		C				0.075	0.250	0.2500	0.0875
		R			0.182				
1600	580	C				0.250			0.1190
1680	638								0.2500

[a] All measurements except total subsidence (0.250 in.) are an average of four readings. Magnification 16 to 1.

[b] Revolutions per minute and model ratio are related by the following formula:

$$Q = \frac{4\pi^2 r n^2}{g 60^2}$$

where Q = model ratio, or that number by which the linear dimensions of model are multiplied to give similar dimension of prototype represented,
n = revolutions per minute obtained with speed counter,
r = radius of rotation in feet, measured in centrifuge,
g = gravity = 32.2,
π = 3.1416.

[c] L = left center; R = right center; C = center. For illustrations, see Figs. 6 to 12.

This may be illustrated by comparing the behavior of a simple beam with a fixed beam. In a simple beam the ends over the supports are free to move; in a fixed beam they are held solidly in place, and as the beam deflects, pressure is exerted at the supports. For a simple beam uniformly loaded the maximum positive moment $M = \frac{WL^2}{8}$ at the center, where W = load per unit length and L = span. There is no negative moment. For a beam fixed at both ends, of the same dimensions as the

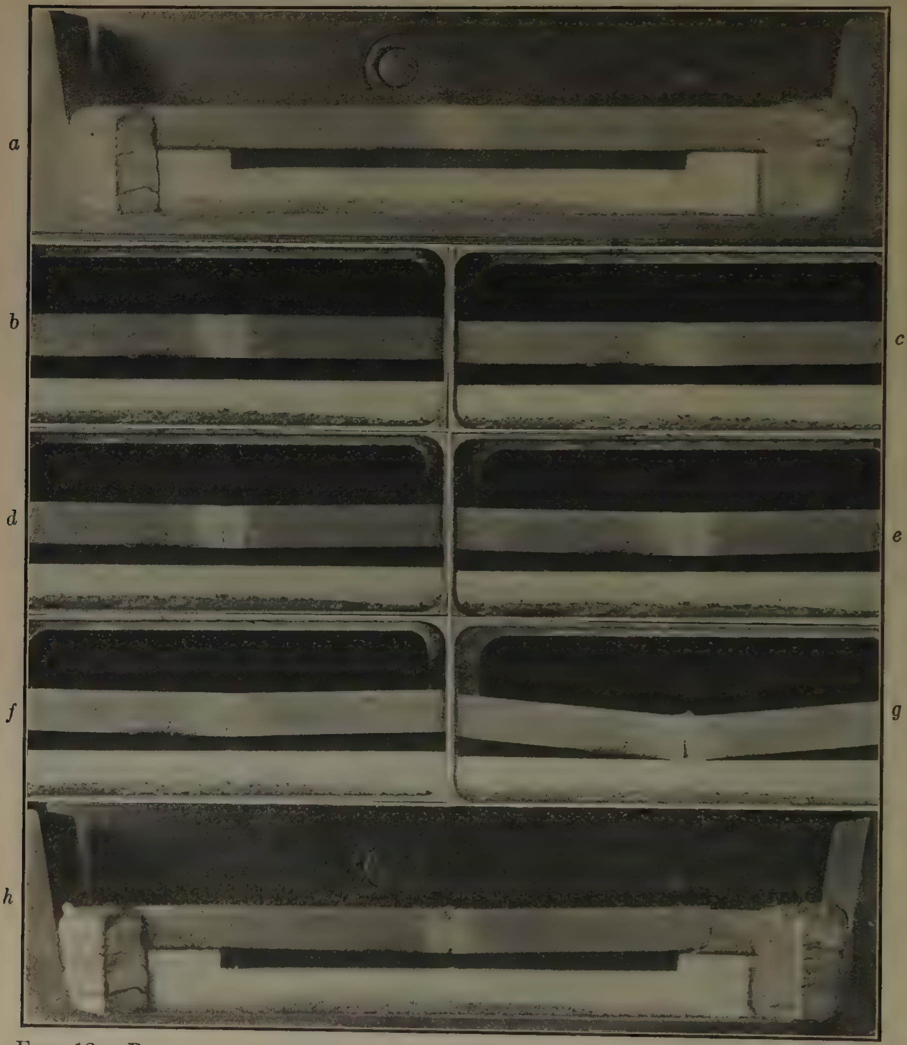


FIG. 12.—PHOTOGRAPHIC RESULTS OF GROUTED ROOF-CRACK TESTS, MODEL 15G

a. Before placing in centrifuge.
b.–g. In centrifuge, while running.
b. 400 r.p.m., 36.3 model ratio
c. 900 r.p.m., 183 model ratio
d. 1200 r.p.m., 326 model ratio
e. 1400 r.p.m., 444 model ratio
f. 1500 r.p.m., 510 model ratio
g. 1680 r.p.m., 638 model ratio
h. After breaking.

simple one and uniformly loaded, the maximum positive moment is $\frac{WL^2}{24}$ at the center and the maximum negative moment at the supports is $\frac{WL^2}{12}$. The maximum unit stresses developed are given by the formula $S = \frac{MC}{I}$, where S = maximum unit stress, M = moment at that section, C = distance from remotest part to neutral axis and I = moment of inertia. Since C and I are the same for both beams, constant K may be substituted for $\frac{C}{I}$ and the formula for maximum unit stress now becomes $S = MK$. Since M for a simple beam is three times the positive moment of a fixed beam $\left(\frac{WL^2}{8} \div \frac{WL^2}{24} = 3\right)$ and 1.5 times the negative moment $\left(\frac{WL^2}{8} \div \frac{WL^2}{12} = 1.5\right)$, the simple beam will break with a shorter span than a fixed one because the maximum unit stresses at bottom and top center of a simple beam will be three times those of a fixed beam for the same span. The fixed beam will have its greatest stress over the support, so that with a fixed beam cracks would be expected to open on the surface near the supports, while the roof underground looked good. This is assuming the roof stratum solid and extending to the surface.

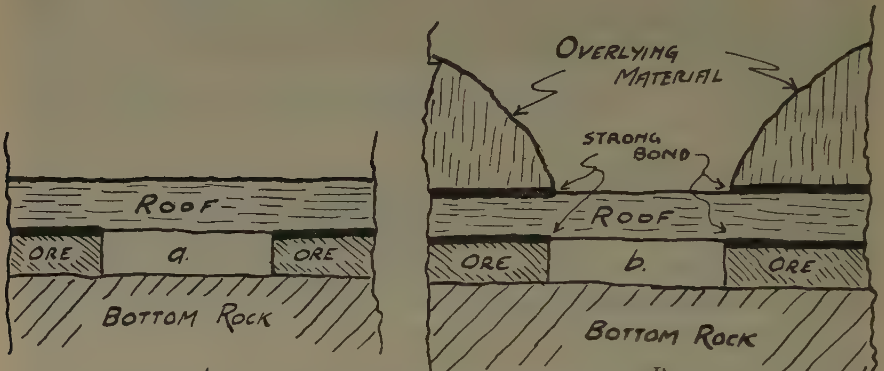
The factors tending to fix the roof beam are: (1) the bond between the roof stratum and the strata immediately above and below; (2) the extent of these strata on either side of the opening; (3) the vertical extent of cover over the strata immediately away from the opening; (4) the structural characteristics of the overlying and underlying material, especially with regard to density modulus of elasticity and unit strengths. This is illustrated in Fig. 13. If openings *a* and *b* are the same and the roof over opening *a* stands, then the roof of opening *b* will stand, will deflect less, and have less stress in it than *a*. The degree of fixation of the roof *a* is very small, while the degree of fixation of *b* is large.

It should be pointed out that the ability of a beam to support itself over a span does not increase indefinitely as the degree of fixation is increased. The maximum is reached when the degree of fixation is such that the unit stresses developed by the roof beam over the span or supports are sufficient to cause it to fail.

EFFECTS OF NUMBER AND POSITION OF CRACKS

From Figs. 8 to 11 and Table 1, in comparing the behavior of models 15C, 15E and 15F, it is interesting to see that while the number and character of cracks differed they all broke at the same number of revolutions per minute (1500). The points in common of these three models were: (1) the thickness of cover; (2) the span; and (3) the fact that the two sides of all cracks were in juxtaposition. From this it appears that the number and position of vertical cracks do not materially affect strength.

In comparing subsidence data, the solid beam 15D has less vertical subsidence than any of the others (except 15G at 1600 r.p.m.) at equivalent model ratios. In comparing 15C and 15E, it is reasonable to believe that the smaller amount of subsidence in 15E is due to the fact that closer contact was obtained between the sides of the crack. For 15C, the ends in juxtaposition were flat-surfaced on a grinding machine and painted with lampblack. This gave a smooth, well lubricated joint, while the cracks in 15E and 15F were jagged and placed in close contact. The subsidence of 15F (Figs. 10 and 11 and Table 1) is greater at the same model ratio than 15E. Since 15E has a crack at the center and 15F has none, it may be reasoned that from the standpoint of vertical subsidence a crack in the roof center with sides in juxtaposition will cause less vertical subsidence than one at a distance from the center. With a center crack both cantilevers have the same deflection at the same time, act on each other, and therefore support each other. Where one cantilever is longer than the other, the longer will deflect the more, and if the inclination of the crack is slightly off vertical and in the proper direction it will deflect or subside much more than the other one. This explains why, in Figs. 6 to 9, one side subsides more than the other.



A B

Fig. 13.—Factors of roof-beam fixation in mining.
Areas of bond contact shown by heavy lines.

The strongest model was the beam with the cemented crack, 15G. It broke at 1680 r.p.m., or a model ratio of 638. The solid beam 15D broke at 1600 r.p.m., or a model ratio of 580. The cracked models 15C, 15D and 15E broke at 1500 r.p.m., or a model ratio of 510. For mining purposes the cracked roofs were nearly as strong as the solid roof and would support openings of approximately the same span as the solid roof.

It is interesting to observe from Figs. 6 to 9 the amount of spring or elasticity present in the model after it is removed from the centrifugal field. At the breaking point in each of the photographs shown, the roof and bottom meet. When the centrifuge speed is reduced, the roof springs back. This residual elasticity would indicate, apparently, that materials stressed beyond the elastic limits have changes of properties which for purposes of calculation would make all the constants differ and act like some new material. Before the use of the present stroboscopic light and photography, this total deflection could not be measured, as subsidence measurements were made in a stationary device after the model was taken from the centrifuge.

CEMENTING OR GROUTING ROOF CRACKS

The roof of model 15G had a crack 0.05 in. wide, the same width as that of 15A. This crack was filled with neat cement. Fig. 12 and Table 1 indicate that it was the strongest of all the models, even stronger than the model with the solid roof, 15D. Model 15G broke at 1680 r.p.m., or a model ratio of 638; whereas 15D broke at 1600 r.p.m., or a model ratio of 580. The similar model without cement broke at a model ratio of 111. Grouting or cementing the crack of 15A made the open span that could be worked underground much greater than that of the span with the open crack. The span that could be worked with the crack grouted, as compared to that of the solid roof, would also be greater. The above shows that grouting or filling roof cracks tends to strengthen mine roofs of this material. The qualiffcations of the grouting material are: (1) a tensile and compressive strength equal to or greater than the roof material; (2) a bond strength between it and the roof material equal to or greater than the tensile and compressive strength of the roof material.

If filling material of this kind is placed in a roof crack at a section of maximum stress, for example, a center crack in a beam, it must strengthen the entire beam; *i.e.*, make the beam (in this case the roof) stronger than if it were a solid one.

The theoretical reasoning may be summed up as follows. In Fig. 14, let BB' be a beam which has assumed its deflection. Then at the center section AA' the stresses are distributed as shown in Fig. 14*b*, AC represents the maximum compressive stress at the top and $A'T$ represents the maximum tensile stress at the bottom. N is the neutral axis or position of no stress. Assume the beam material of such strength that $A'T$ and AC are a little greater than the ultimate tensile and compressive strengths of the roof materials. Cracks therefore occur at A and A'. On nearing the supports S and S', the tensile and compressive stresses in the beam decrease and no cracks appear. Now assume a crack is present at AA' and that it is filled with a cementing material with the following qualifications: (1) Its ultimate tensile and compressive strengths are greater than those of the beam material; (2) the strength of bond in tension and compression is greater than the strength of the beam material. If this filling is placed in the crack, the beam will not fail, because the stresses developed will be insufficient to cause rupture. It is evident, therefore, that a means of overcoming some of the difficulties due to roof cracks, whether in a mine or tunnel, is at hand. Investigation and research on grouting materials and methods of application should probably be carried on.

It may be pointed out here that the general laws of mechanics check the laboratory results and give additional evidence of the validity of this type of research.

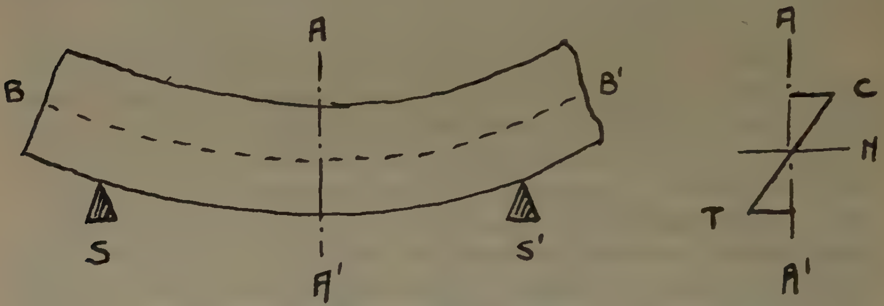


FIG. 14.—DISTRIBUTION OF BEAM STRESSES AT A SECTION.

SUMMARY AND CONCLUSIONS

To the numerous questions as to the effects of cracks in the mine roofs, the following is presented as relating to the effects of approximately vertical cracks in solid horizontally lying roofs of sandstone, as here tested. It is reasonable to assume that the results may be applied to other materials lying in the same position. A paper is now being prepared treating of horizontal cracks, *i.e.*, roofs of bedded materials. For inclined roofs and cracks a larger centrifuge than is now available is necessary.

The results of experimentation may be summed up as follows:

1. Approximately vertical cracks of any number and position, where the sides of all cracks are in juxtaposition, decrease the roof strength by a maximum of less than 15 per cent, roof strength being here defined as directly proportional to the load it will support.

2. The presence of these cracks appears to increase the surface subsidence over that with a solid roof. A center crack causes less subsidence than one off center.

3. Roof cracks of which the sides are not in juxtaposition materially weaken roof strength.

4. Grouting roof cracks whose sides are not in juxtaposition may strengthen the roof to a point equal to or greater than that of a solid roof, depending on the grouting material, its strength, and the bond strength between it and the roof material.

5. An important factor governing roof strength is the degree of fixity of the roof at and beyond its supports. With solid roofs having a high degree of end fixity surface cracks will tend to appear over the supports before they appear underground.

6. The laws of mechanics may be applied to the scientific investigation of mining structures and to their design.

ACKNOWLEDGMENTS

To Dr. T. T. Read and Prof. Robert Peele the author is indebted for much helpful advice, especially in preparation of this manuscript. Mr. G. W. Fiske built all machinery and prepared the models. Mr. L. S. Baldin was the general assistant in the experiments and on him devolved most of the laboratory work and calculations. Dean J. W. Barker furnished helpful suggestions regarding the electrical part of the equipment. To Mr. J. B. Russel, Jr., of the Electrical Engineering Department, to the Sperry Gyroscope Co. through Messrs. Willis and Bassett, and to Mr. H. E. Edgerton, of the Massachusetts Institute of Technology, the author is indebted for the use of Mr. Edgerton's stroboscopic mercury arc light and the correct assembling and operation thereof. Prof. G. B. Pegram of the Department of Physics has continued to give valuable suggestions and much of his time.

DISCUSSION

(George S. Rice presiding)

H. BRIGGS,* Edinburgh, Scotland (written discussion).—Having read Professor Bucky's previous papers dealing with the behavior of models of mine-roof structures subjected to centrifugal force, I am much interested in this last communication in which certain conclusions of a practical character appear.

Experiments with models have been carried out by a number of investigators concerned with mining subsidence problems—myself among them. Some, recognizing the discrepancies between the effects on the natural scale and those displayed by the models, have been cautious in their attempts to elucidate the behavior of rock masses by aid of the latter; others have trusted the models too far, and have been responsible for theories consisting of half-truths that are proverbially hard to destroy. But, so far as I am aware, none of these experimenters realized the need to increase the gravity-equivalent in an inverse ratio to the diminution in scale of the model; this, and the means adopted by Professor Bucky, are new to the subject, and provide a promising line of research.

The large increase in strength resulting from grouting roof cracks is important. Though the method is hardly applicable at faces where coal is being worked by the longwall system (the prevailing method in Great Britain), owing to the rate of advance of the face, it may be practicable in roads, and any additional information as to the most suitable means of applying the cement in practice would be useful. The fact that a vertical crack with its sides close together diminishes the strength by less than 15 per cent is distinctly unexpected, though it goes a good way towards explaining how it comes about that cracked roofs underground frequently stand so well.

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* Professor, University of Edinburgh.

That there should be sufficient residual elasticity in a roof that has broken at the middle of a span to cause the two halves to spring upwards when the load is released (p. 225) is another matter of much interest. One is induced to inquire whether this behavior is due to the heterogeneous character of rocks. Is it possible for the elastic limit to be exceeded in one or more of the minerals of the rock, while other minerals preserve their elasticity, or for the cementing material, say, of a sandstone to be strained beyond the limit and the quartz grains not? If a homogeneous beam is tested, for instance, one of cement, is the same effect apparent?

It may not be out of place to observe that the orthodox diagram of a beam and the stresses operating in it (Fig. 14) have little or no resemblance to a mine roof spanning an opening. The development of compression in the upper part of a beam cannot occur unless the ends have freedom of lateral movement. In a sagging roof span, where the ends are rigidly gripped, the prevailing stress is tension throughout the "beam" with the exception of a short portion on the underside near each end, and the neutral axis lies for the most part outside the "beam."

I hope Professor Bucky will be able to continue his important and novel researches.

G. S. Rice,* Washington, D. C.—The U. S. Bureau of Mines has shown that the strength and compressibility of the buttresses are all-important in the development of the full strength of monolithic concrete stoppings underground and this would undoubtedly also be true of natural flat arches. In reference to the grouting of cracks in the roof, those cracks sufficiently open for grouting to enter would be tension cracks and there surely would not be any gain in tensile strength by filling them with cement. The grouting, however, would serve a valuable purpose in excluding the moisture and air and tend to prevent oxidation, which has so important an effect in weakening the strength of some rocks, especially shales.

H. G. Moulton,† New York, N. Y.—In the case of multiple vertical cracks, the cantilever effect stops at the crack nearest the supporting pillar. If some settlement takes place the entire action becomes that of a flat arch.

R. D. Hall,‡ New York, N. Y.—What is the difference between a beam and an arch? They are subject to the same stresses; compression in the top and tension in the bottom. Thus no distinction between them is warranted.

G. S. Rice.—The U. S. Bureau of Mines in tests made in conjunction with the Bureau of Standards found that certain flat slabs of concrete, supported as simple beams and uniformly loaded (by compressed air), ruptured at 10 lb. per square inch but when their *ends were restrained* and they acted as flat arches they did not fail until a pressure of 120 lb. per square inch was reached.

Mr. Hall's statement might be misunderstood, as the term beam is usually understood; namely, without restraint at the ends. However, if it does have end restraint, as a roof slab usually does, it then has all the effect of a flat arch, and assuming the side walls were strong, failure usually occurs by crushing at the top of the roof slab or layer at its central part. If there is a natural fracture or joint plane, however, at one side or the other of the heading, entry, or room, failure would likely occur by slippage or shear.

W. R. Crane, Washington, D. C. (written discussion).—The effect of continuity of beds is a vital matter relative to ability to support themselves and the overlying and largely fractured beds. As Mr. Moulton points out, a cantilever exists only from the support to the nearest crack; in fact, as joints and slips occur comparatively close together, averaging close to 24 in., the cantilever action is reduced to a practically negligible quantity. The strength of beds broken at frequent intervals by planes of weakness is due almost entirely to the compressive strength of the rock at the top of the joint and slip planes.

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* Chief Mining Engineer, U. S. Bureau of Mines.

† Consulting Mining Engineer.

‡ Engineering Editor, *Coal Age.*

It would seem more desirable to study the failure of the immediate roof of mine excavations by observations of actual conditions of occurrence in the mines than to test small pieces of rock that lack the physical characteristics of beds in place.

Further Notes on Bumps in No. 2 Mine, Springhill, Nova Scotia

By T. L. McCall,* Sydney, N. S.

(New York Meeting, February, 1934)

Abstract†

This is a continuation of the paper on bumps presented by the late Mr. Herd in February, 1929. The seam worked consists of bituminous coal, 9 ft. thick, pitching 20°, and now under a depth of cover of 2800 to 3200 ft. Full particulars are given of two bumps that happened in November, 1931, with an interval of only 9 hr. between them. The first was a "district" bump affecting an area of 1000 ft. on the strike by 700 ft. on the dip and rise, beginning at a distance of 1000 ft. from the longwall faces. The second, which caused much greater damage, was localized to two levels 200 ft. apart, in the area left undisturbed by the first bump next to the longwall faces.

Observation stations had been established in the workings in 1931 and definite records were obtained on the flow of coal taking place from the edge of large pillars developed for retreating work, into the open roadways. The records also gave evidence that "advance weighting" or "underdraw" was affecting the coal at an average distance of 260 ft. in advance of the longwall faces.

The author propounds the theory that the sandstone bed varying from 0 to 75 ft. in thickness and lying 14 ft. above the coal acts as a slab in distributing the superincumbent weight. The weight is thereby concentrated on the pillars in advance of the longwall faces and this load, in conjunction with a pavement strong enough to resist the reaction, leads to overstressing the pillars, which finally collapse suddenly, thus giving rise to this particular type of low side bump. The overlying sandstone bed, which is apparently a condition common to all coal mines in which bumps occur, is not necessarily a massive bed, as the thickness of this bed at Springhill in the area that has experienced these severe low side bumps is only 25 to 31 feet.

A list of 43 minor bumps occurring in the past 2½ years is appended to the paper.

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[*] Chief Mining Engineer, Dominion Steel & Coal Corporation.

[†] This paper, with discussion, appears in *Trans.* A.I.M.E. (1934) **108,** Coal Division.

Experimental Air-conditioning for the Butte Mines

By William B. Daly* and A. S. Richardson,† Butte, Mont.

(New York Meeting, February, 1934)

The application of artificial refrigeration, or air-conditioning, to the ventilation of deep, hot mines has long been a subject of interest to the operators of such properties. Artificial cooling of the air in theaters, railroad coaches, and various industrial plants has been so successful within recent years that it has been natural to assume that the adoption of similar methods would afford a ready solution to the difficult problem of mine ventilation. It should however, be obvious, that although there may be nothing technically impossible in the application of this process to mine ventilation there is a great economic distinction between the maintenance of a high standard of comfort for a large number of people in a relatively small space such as a theater and the cooling of air to improve working conditions for miners scattered throughout the extensive workings of a large mine. Many methods of air-conditioning have been considered by the management of the Anaconda Copper Mining Co. at various times, but until the development of the process to be described in this paper none has seemed economically practicable.

The Ventilating Problem at Butte

The main obstacle to satisfactory ventilation in the Butte mines is the normally high temperature of the granite country rock. In the present operating zone, rock temperatures as high as 120° F. have been noted, and in some parts of the district the rate of increase of temperature with depth has ranged as high as 3.5° F. per 100 ft. Other factors, such as the decay of mine timber and the oxidation of sulfide minerals in the ore bodies, contribute to raise the temperature of the air in the working places but, particularly in the development of the mines to greater depth, the naturally high temperature of the rock is of predominant importance.

In few other places, probably, are the ordinary ventilating measures more hindered than at Butte. At an operating depth now exceeding 3000 ft., all of the air required for ventilation must pass through timbered shafts which, because of heavy ground, are of small size and offer great resistance to the flow of air. Shaft compartments are rarely more than 4.5 × 5 ft., and the shaft area open to the flow of air usually does not exceed 65 sq. ft.; in only two shafts is the area as great as 85 sq. feet.

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* Manager of Mines, Anaconda Copper Mining Co.

† Ventilation Engineer, Anaconda Copper Mining Co.

A further difficulty arises from the heating of the air as it flows through the inlet shafts and before reaching the working places. Starting at −20°F. at the surface, the air usually attains a temperature of more than 65°F. before leaving the shafts at stations below the 3000-ft. level. While traversing the connections between the shafts and the mining areas on the veins, or in crosscuts and drifts, there is a further increase ranging from 5° to 15°F., so that the air loses much of its capacity for cooling rock surfaces before it reaches the working places.

In general, the mechanical ventilation of the Butte mines employs an exhaust system of which the operating shafts are the inlets. In them the air velocity does not usually exceed 1200 ft. per min., but in the outlet shafts, at the tops of which the main reversible fans are installed, the air velocity ranges from 2000 to 3500 ft. per min. in most places. To reduce the resistance at these high velocities, practically all of the outlet shafts have been surfaced, so that each compartment is a separate smooth-surfaced air duct. When extending these shafts to greater depths, an octagonal cross-section has been substituted for the original two or three rectangular compartments.

The main surface fans, with one exception, are of the reversible centrifugal type and develop pressures ranging from 3.5 to 7 in. of water. Wherever greater pressures have been required, underground booster fans have usually been installed, which also assist in the distribution of air to the working places. One recently installed surface fan has been of the propeller type, exhausting 300,000 cu. ft. of air per min. against a static pressure of 9 in.; this has permitted a notable increase in circulation and the elimination of a number of booster fans. For the ventilation of dead ends, and other places where the general circulation does not supply sufficient air, a number of small blowers are used in conjunction with flexible tubing.

When operating at full capacity, the total volume of air in circulation approximates 3,000,000 cu. ft. per min. There are 24 main reversible surface fans, about 20 underground booster fans, and 450 auxiliary ventilation blowers equipped with 25 miles of flexible tubing. The total power consumption for ventilation approximates 8500 horsepower.

Outline of the Cooling System

In spite of the heavy expenditures already involved, it has for some time been problematical whether ordinary methods would be able to cope with the conditions that will be encountered at still greater depths. Fortunately, however, climatic conditions in the district, where low temperatures prevail during the greater part of the year and humidity is very low during the summer, provide a natural source of cooling effect which has almost infinite capacity. This natural advantage is utilized by the existing mechanical system to some extent, but only incompletely. This cooling power of the surface atmosphere being available for air-conditioning, only a workable plan was needed for its utilization, and this has now been devised.

The essential feature of this plan is the circulation of calcium chloride brine in a closed circuit in which the brine is alternately heated by the warm air within the mine workings and cooled by the cold air of the surface atmosphere; it thus serves continuously to cool the warm mine air and to dissipate the heat at the surface. The main items of the equipment are illustrated diagrammatically in Fig. 1, and include:

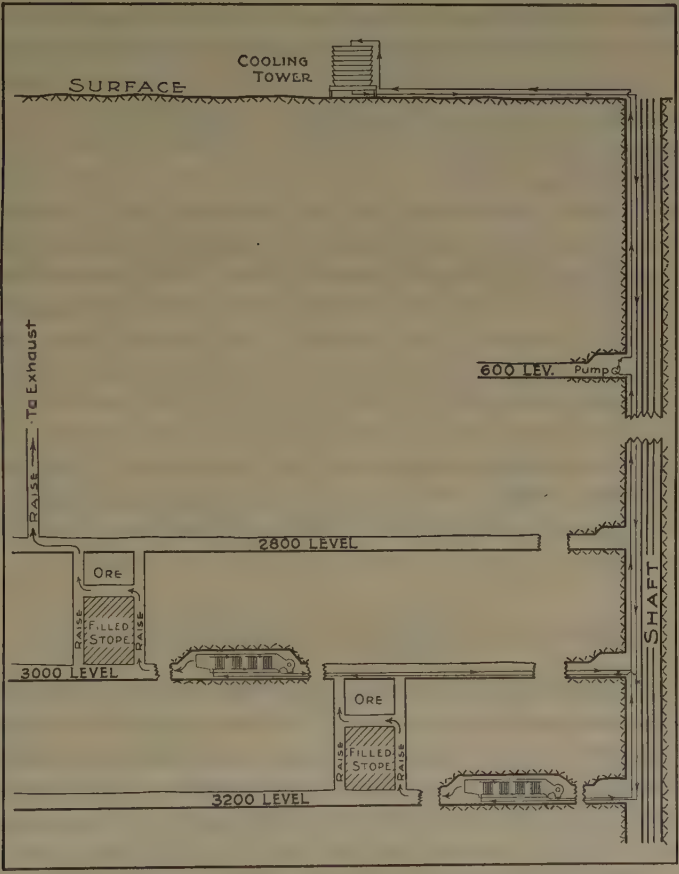

Fig. 1.—Diagram of the cooling and ventilating system.

1. A cooling tower at which the brine is cooled by the surface atmosphere. Temperatures as low as −20° F. are not uncommon in the Butte district during the winter, and the wet-bulb temperature seldom exceeds 60° F. during the summer.

2. Pipe lines to carry the brine in closed circuit. The hydrostatic head in the two lines being in balance, power is required only to overcome frictional resistance, plus the small lift to the top of the tower.
3. Main brine circulation pump.
4. Mine air-conditioning plants. Brine from the main inlet circulates through high-pressure pipe coils which complete the closed circuit to the return line. Water is pumped through sprays into the warm mine air and falls on to the pipe coils, thus transferring heat from the mine air to the brine. The spray water then falls into a sump from which it is again pumped to the sprays, completing its own closed circuit. Circulation of brine through the pipe coils is in a direction opposite to the air flow.

Brine may thus be delivered at the air-conditioning plant, through an insulated pipe line, at a temperature not more than 5° F. higher than that at which it leaves the surface, and the mine air may, if necessary, be cooled to a temperature only about 8° F. higher than that of the brine. Since a cubic foot of brine will absorb about 4000 times as many heat units as a cubic foot of dry air, through the same rise in temperature, it is evident that a relatively small pipe will carry enough brine to dissipate more heat than could be dissipated by an air flow which would require two or more large air shafts. In most mines the ventilating current removes a considerable quantity of water in the form of vapor so that the latent heat of evaporation represented in the increased moisture content of the air is of considerable importance and greatly reduces the ratio of effect that has been mentioned. However, the facts stated afford a rough comparison of the possibilities of the two methods.

Preliminary Experiments and Observations

Experimental development of this plan was begun in December, 1930, and has been in progress ever since. The first step was the construction of a surface plant at which air might be heated and humidified so as to duplicate mine conditions, and then be subjected to the air-conditioning process. The practicability of the method was thus demonstrated, and the engineering data necessary for the design of such plants were obtained.

An underground plant was next installed on the 3200 level of the Mt. Con mine, also of an experimental character. To avoid the expense of the high-pressure pipe lines required for closed circulation of the brine, it was decided to use water from the city supply system, pre-cooled so far as possible at the surface, and to discharge it into the drainage system on the 2800 level after use; such an expensive method would, of course, be permissible only for experiments or as a temporary expedient. The plant was operated, mainly, to obtain data on its effectiveness in reducing temperature and humidity in the working places and also to determine the effect upon the ground and the mine timbering. Opportunity was also taken to make a number of tests on the suitability of various materials to resist corrosion.

The results attainable by this process are wholly dependent upon the temperature and humidity of the surface atmosphere. In addition to our general information, more exact data were obtained by dry and wet-bulb recording thermometers; readings of the recording instrument were checked at intervals against a sling psychrometer. The record for an entire year is given in Table 1, showing the range of conditions for each month; wet-bulb temperatures below the freezing point have been omitted because they are of doubtful accuracy and are not important.

Table 1.—*Dry and Wet-bulb Temperatures*

	Temperature, Deg. F.			Wet-bulb Temperature, Deg. F.		
	Mean Minimum	Average	Mean Maximum	Mean Minimum	Average	Mean Maximum
November, 1931	7.3	17.9	35.2			
December	12.9	20.5	41.7			
January, 1932	5.9	12.0	32.6			
February	15.8	23.0	40.0			
March	14.3	19.9	35.2			
April	26.2	35.1	47.4			
May	34.2	45.9	59.8			
June	42.8	56.0	70.4	39.4	46.2	54.3
July	49.6	62.3	78.0	44.0	51.5	57.4
August	46.8	61.5	78.1	41.7	50.0	58.2
September	42.2	53.0	71.9	35.9	44.2	54.9
October	27.5	36.1	46.4			
Yearly average	27.1	36.9	53.0			

Conditions affecting the operation of the plant during the summer months are further illustrated in Fig. 2, which shows the variations of the different factors during the 24 hr. of a clear day. Only two basic factors are important in this connection, the temperature and the water-vapor content of the air; the others, such as relative humidity, wet-bulb temperature, and dew point, are functions of the first two variables.

When cooling brine, or water, by contact with the air, as in ordinary cooling tower practice, the theoretical low limit is, of course, the wet-bulb temperature. During the warmest part of the day, the wet-bulb temperature is usually from 15° to 30° F. lower than the sensible temperature, and the dew point is about 10° to 20° F. lower than the wet-bulb temperature; even during July and August, at Butte, the dew point frequently drops below the freezing point.

A good illustration of the heating of air as it descends in an inlet shaft is shown in Table 2, recording some data obtained at the Berkeley shaft, which has not been used for operating purposes for a number of years. The velocity of air is approximately 900 ft. per minute throughout the whole depth of the shaft and temperature is unaffected by any re-entrant air. The two sets of readings closely represent the difference between winter and summer conditions, as they were taken late in February and early October.

TABLE 2.—*Air Conditions at Berkeley Shaft*

	February		October	
	Collar	2800-ft. Level	Collar	2800-ft. Level
Barometer, in. of mercury	24.15	26.16	24.07	26.14
Temperature, deg. F.	34.5	60	49	69
Wet-bulb temperature	34.5	55	44	61
Humidity, relative, per cent	100.0	74	69	64
Humidity, grains water vapor, per cu. ft.	2.322	4.251	2.716	4.944
Density of air, lb. per cu. ft.	0.0647	0.0668	0.0625	0.0652
Theoretical temperature due to compression, deg. F.		46		61

Increase in heat per 10,000 cu. ft. of air at 2800 level:

	B.t.u.	Per Cent	B.t.u.	Per Cent
Theoretical heat of compression	1,785	26.6	1,804	29.2
Latent heat of evaporation of increased moisture	2,752	41.0	3,157	51.1
Increase in sensible heat of air and vapor	3,969	59.0	3,018	48.9
Total increase	6,721	100	6,174	100

The increase in temperature and humidity of the air flowing through the Berkeley shaft is somewhat lower than usual; the shaft bottom is at the 2800-ft. level.

More complete data on the amount of heat gained by the air in deeper and more extensive workings were secured at the Mt. Con mine, where the experimental air-conditioning plant was installed; here the stoping area is practically all below the 2800 level. The figures are in Table 3.

DESIGN OF EXPERIMENTAL AIR-CONDITIONING PLANT

Plans for the experimental conditioning plant were based on a nominal capacity for cooling 15,000 cu. ft. of air per min. from an initial wet-bulb temperature of 80° F. down to a final wet-bulb temperature of 50° F.,

TABLE 3.—*Heat Gained by Air at the Mt. Con Mine*

	Barometer	Temp., Deg. F.	Wet-bulb Temp., Deg. F.	Grains Water Vapor per Cu. Ft.
Collar of inlet shaft	23.77	57	47	2.4
3100 station of inlet shaft	26.36	65	59	4.8
3100 level, crosscut at vein		71	64	5.8
Discharge to outlet air shaft		87	82	11.2

Heat gained per 10,000 cu. ft. of air measured at surface:

	B.t.u.	Per Cent
Heat gain in inlet shaft	3,820	23
Heat gain in crosscuts to vein	2,180	14
Heat gain in working zone	10,300	63
Total heat gain in mine	16,300	100

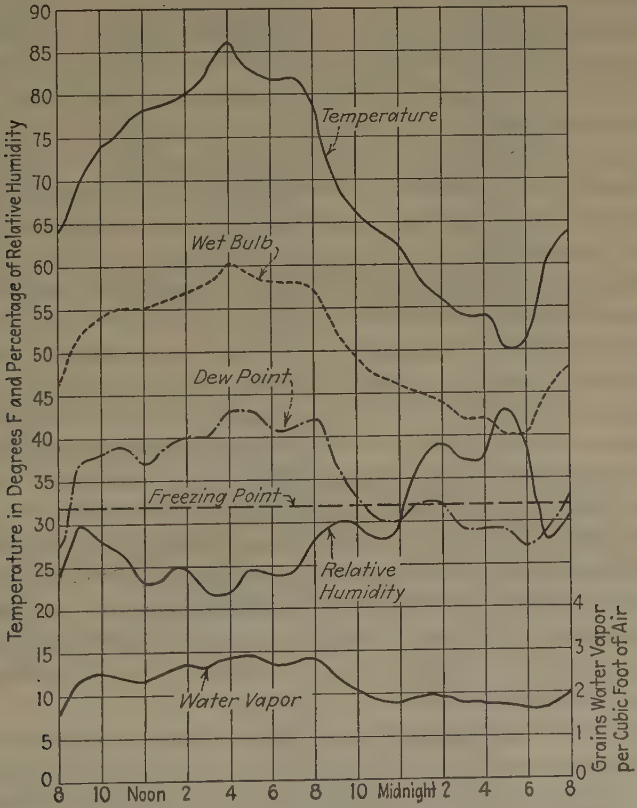

FIG. 2.—RECORD FOR JULY 7–8, 1932, AT BUTTE, MONT. BAROMETRIC PRESSURE, 25 INCHES.

with 90 gal. of brine, or water, per min., at a temperature of not over 40° F. It was recognized that the temperature of the brine supply would rise considerably above 40°, and that the effect of variations in the volume and final temperature of the conditioned air should be taken into consideration. A fan, therefore, was installed which would, if necessary, increase the flow of air to approximately 30,000 cu. ft. per min., and the conditioning plant was designed to obtain maximum benefit from the limited and expensive water supply.

The main feature of the plant was a wooden box, about 4 ft. wide and 7 ft. high inside, and 55 ft. long, through which the air to be conditioned was circulated; in it were placed the pipe coils and sprays to operate in the manner already described. The pipe coils were installed in four groups in series, each group having from 9 to 15 coils in parallel for testing under different conditions of operation. The pipe was ¾ in. extra heavy, and the materials were copper, brass, galvanized steel, wrought iron, steel, and copper-finned tubing. Each of the four groups of coils was served by a separate pump circulating the spray water in closed circuit.

When operating such a plant, it is possible to circulate the conditioned air continuously in a closed circuit through the workings to be ventilated and back to the plant, or to take air reaching the plant from some other part of the mine and, after having conditioned it, to circulate it forwards through the ventilation system towards the outlet air course. Under the first method of operation, the velocity of air movement, or the volume of air passing through the conditioning plant, may be indefinitely increased, subject only to the frictional resistance of that section of the mine affected and to the necessity for adding sufficient fresh air to maintain a satisfactory proportion of oxygen. This method has certain advantages in that it is possible to increase considerably the effective cooling power of the air, at slight expense, by increasing its velocity, and also to abstract the desired amount of heat by cooling a larger volume of air through a smaller temperature range, an arrangement which contributes materially towards plant efficiency. The second method of operation limits the volume of air treated to that which can be spared for the ventilation of that particular section of the mine, and the required cooling effect must be obtained by cooling that volume of air to a temperature low enough to accomplish the desired result. In practice, the method selected will depend largely upon mine conditions, and is most likely to be some compromise arrangement not closely restricted to the limitations of either method.

In this experimental work it was first intended that the conditioned air should be circulated mainly in closed circuit and should be utilized for the ventilation of not more than three or four working places. This arrangement was adhered to for a length of time sufficient to observe

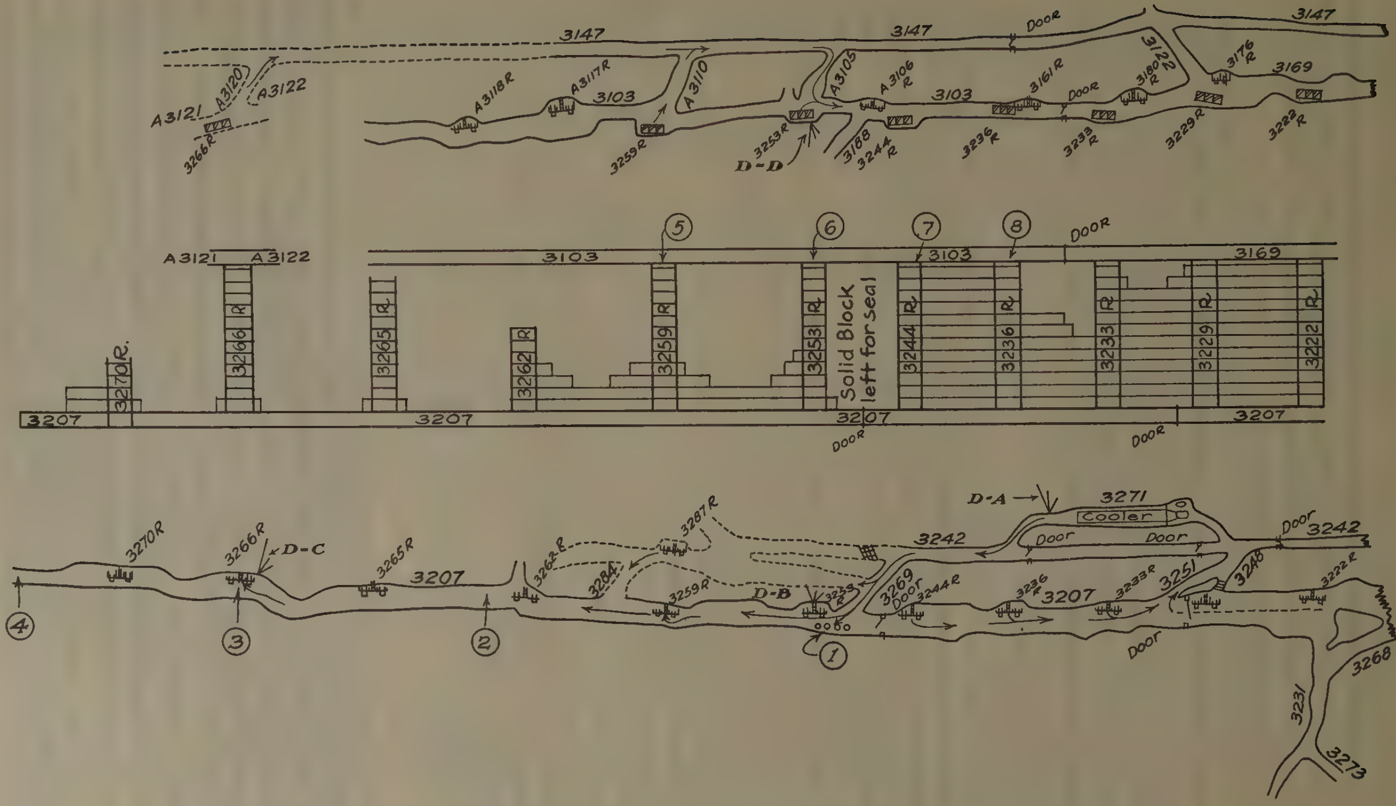

FIG. 3.—ARRANGEMENT FOR EXPERIMENTAL AIR-CONDITIONING AT THE MT. CON MINE.

important changes in conditions, and the plant was then utilized to give the maximum benefit to a rapidly increasing number of working places. The area ventilated by the plant was then considerably greater than that for which it was intended, and subsequent changes in ventilation conditions were considerably affected.

TABLE 4.—*Operating Results of Experimental Air-conditioning Plant*

Data	Oct. 22, 1931	Jan. 30, 1932	Mar. 20, 1932	Mar. 29, 1932
Air volume, cu. ft. per min	18,200	17,000	30,000	16,000
INTAKE AIR:				
Temperature, deg. F	82	83	81	83
Wet bulb, deg. F	80	76	77	76
Humidity, per cent	92	74	84	73
Humidity, grains vapor per cu. ft.	10.7	8.87	9.47	8.75
DISCHARGE AIR:				
Temperature, deg. F	65	54	62	46
Wet bulb, deg. F	65	54	62	46
Humidity, per cent	100	100	100	100
Humidity, grains vapor per cu. ft.	6.78	4.68	6.14	3.54
BRINE OR WATER SUPPLY:				
Gal. per min	90.3	94.5	91.1	92.8
Intake temperature, deg. F	54	41	38	34
Discharge temperature, deg. F	76	64	69	62
MOISTURE REMOVED FROM AIR:				
Percentage of initial content	36.5	47.2	35.1	60.0
Gal. of water per min	1.22	1.22	1.72	1.44
COOLING EFFECT, B.T.U. PER MIN.:				
Sensible cooling effect	5.680	7,240	7,340	8,920
Sensible cooling effect, per cent	37	40	31	41
Condensation	10,800	10,800	16,100	12,650
Condensation, per cent	63	60	69	59
Total cooling effect	16,480	18,040	23,440	21,570
Equivalent tons of ice melted per 24 hr.	82	90	117	107

The location of the conditioning plant in reference to the workings is shown on Fig. 3. Originally, the main courses of air circulation were from the plant on the 3200-ft. level through lateral and crosscut to 3207 drift west, and up 3253 and 3259 raises to the 3100 level. A solid block of ground was purposely left between 3244 and 3253 raises, and a door in the drift between the bottoms of these raises confined the air to this course. On the 3100 level the greater part of the air flowed east to 3244 and 3236 raises, through which it returned to the 3200 level and back to the plant. Some air passed forward through 3161 and 3180 raises to upper levels, and an equal volume of air was drawn into the system from workings below and east of 3251 crosscut. At the station marked No. 1, four blowers drew fresh air from the plant and forced it through flexible tubing to dead-end faces west of 3259 raise. Progress of later developments is partly indicated by dotted lines, but raises from the 3300 level were finally holed west of the conditioning plant and recirculation systems were established both upward and downward from the 3200 level.

When considering these data, it should be borne in mind that, with circulation of brine in closed circuit, the temperature and volume of brine reaching the plant may be varied beyond limits attainable in this experimental work. During the winter months it would be possible, if desired, to cool the air to temperatures near the freezing point, and during the summer it is expected that the final temperature of the air will not exceed 65° F., except for a few hours of the day during abnormally warm weather.

EFFECT ON TEMPERATURE AND HUMIDITY

The improved conditions in the working places are evident from the data in Table 5; the stations to which the records refer are those numbered in Fig. 3. Figures for the first date, Oct. 10, represent the conditions existing before the plant was put into operation.

TABLE 5.—*Conditions before and during Operation of Cooling Plant, Temperatures, Degrees Fahrenheit*

Station No.	Oct. 10, 1931		Oct. 22, 1931		Jan. 30, 1932		Mar. 20, 1932	
	Temp.	Wet Bulb	Temp.	Wet Bulb	Temp.	Wet Bulb	Temp.	Wet Bulb
1	85	83	72	70	60	58	65	64
2	93	86	86	82	83	79	79	75
3	92	86	87	83			80	77
4	97	90	93	89	82	76		
5	90	79	83	78	75	72	73	70
6	92	86	79	72	68	65	65	63
7	89	83	77	73	75	72	71	68
8	88	81	84	72	75	72	72	68

Prior to the operation of the plant, the rock surfaces throughout the whole of the area were moist, the running boards in the drift were wet, and also most of the mine timber. A few weeks after operation began practically all free moisture disappeared from the rock surfaces, except at a few places where small water flows occurred, and conditions underfoot were generally dry. Mine timber was also dry, and repair work due to heavy ground and broken timber was greatly reduced.

DUST AND OTHER IMPURITIES

One effect of this air conditioning which has considerable importance in mining operations is the elimination of dust from the mine air. In most cases, when air is treated by water sprays, as in the ordinary air washer, or with water blasts, the humidity of the air is increased and the elimination of dust is limited to coarser particles mechanically carried down by the falling drops of water. In the present process, on the contrary, from 35 to 60 per cent of the moisture originally present in the air is removed, and elimination of dust is accomplished by condensation of the vapor around the dust particles, as nuclei. At each of the four stages of water spraying an accumulation of precipitated dust was noted, progressively decreasing in quantity and in size of particles at each successive reduction in the temperature and the amount of water vapor in the air. Finally, the condensed water, continuously flowing from the plant, carried a certain amount of dust which was finest of all.

Regarding the elimination of carbon dioxide and other objectionable gases, not much can be said since no appreciable quantity of such gases was ever observed in the air. Since one or two gallons of fresh water per minute were continuously added to the spray system, there was always enough fresh water to absorb any carbon dioxide. A few tests showed the presence of 0.04 per cent carbon dioxide in the air before conditioning and none after. About one-half of the powder smoke reaching the plant during blasting periods was eliminated on the first pass, and with the recirculating system of ventilation all smoke was completely removed in about 15 minutes.

Reduction of Rock Temperatures

Since most of the heat in the mine workings comes from the rock, information as to the effect of the conditioning plant on the rock temperatures was thought to be useful. From previous observations it was known that the original rock temperatures were near 100° F. A number of holes, from 5 to 15 ft. deep, were drilled into the rock at points where significant changes in temperature might occur, but certain conditions caused the holes to cave and it was possible to obtain only a few scattered readings. Later, other holes were drilled and cased with pipe, so that some additional readings were obtained, but these holes also were lost before long, so that the record is not altogether satisfactory.

The location of these cased holes is shown on Fig. 3. They were drilled in groups of three, respectively 5, 10, and 15 ft. deep, in order that the temperature of the ground at these depths might be determined at nearly the same time. Readings were obtained by maximum registering thermometers which were placed at the bottoms of the holes and left there for about 6 hr., during which time the end of the hole was plugged tight. The record is given in Table 6.

TABLE 6.—*Temperature of Rock in Drill Holes*

Station	Hole No.	Depth of Hole, Ft.	Date	Temperatures, Deg. F.		Remarks
				Air	Rock	
D-A	1	5	12/28/31	57	84	Cased hole
		3	1/24/32	56	80	Hole caved at 3 ft. depth
			2/ 7/32	57	78.5	Lost hole soon after this date
D-A	2	8½	12/28/31	57	88.5	Depth of hole 10 ft.
		8	1/24/32	56	85.0	Partly caved
		7	2/ 8/32		83.0	
D-A	A	12½	11/ 5/31	67	97.5	No casing in this hole
		12½	11/17/31	56	96.5	
		12½	12/ 1/31	57	94.5	
D-A	3	14	12/29/31	57	94.5	Depth of hole 15 ft., cased
		14	1/30/32	55	92.5	Partly caved
D-B	1	5	12/29/31	69	86.5	Cased hole
		5	2/ 9/32	74	79.5	Hole caved after this date
D-B	2	10	12/29/31	69	87.0	Cased hole. Depth 10 ft.
		10	1/25/32	67	86.0	
		10	2/ 9/32	74	85.0	Hole caved after this date
D-B	3	15	12/30/31	68	90.5	Cased hole. Depth 15 ft.
		9	2/ 5/32	67	85.0	Casing squeezed at 9 ft.
		9	2/ 9/32	74	84.5	Hole caved after this date
D-C	1	5	12/30/31		95.0	Cased hole. Depth 5 ft.
		5	1/26/32		93.0	Raised hole 2/19/32
		5	2/10/32		92.5	Return air movement intermittent
D-C	2	10	12/30/31		95.5	Cased hole. Depth 10 ft.
		10	1/26/32		95.5	
		10	2/10/32		95.5	
D-D	1	5	12/31/31	71	85.0	Cased hole. Depth 5 ft.
		5	1/27/32	69	83.0	
		5	2/ 7/32	68	82.5	
D-D	3	10	1/30/32	69	85.0	Cased hole. Depth 15 ft.
		10	2/ 3/32	68	84.5	Casing tight at 10 ft.
		10	2/11/32	68	83.5	

CONCLUSIONS

The most important fact demonstrated by this experimental plant is that this air-conditioning method permits the temperature and humidity of the air in the working places to be controlled within limits not possible by ordinary ventilating methods. Tentatively it is planned that, as an average mine condition, the temperature of the air shall not exceed 75° F. with a relative humidity of 75 per cent, and the realization of such a standard seems entirely practicable.

As to the elimination of dust from mine air, the results are believed to be better than those obtainable by present methods, such as water blasting, but the drying of the working places by the de-humidification of the air naturally tends to increase dustiness, and an extension of the practice of wetting down the working places will probably be necessary.

From the operation of the plant it was learned that a design providing for very considerable variation in capacity will be desirable. When such a plant is first put into operation the area to be ventilated may be small but it may soon become necessary to ventilate a much larger area. Still later, when most of the ore in that section of the mine has been removed, a smaller quantity of conditioned air may again be sufficient. This flexibility in capacity affects not only the air-conditioning equipment itself but also the selection of suitable fans.

The major item of expense is the cost of the main circuit pipe lines, which will naturally vary considerably at different mines. Assuming that new shafts were to be sunk, crosscuts made to the different veins, and a fan installed, in accordance with present practice, the initial expense would probably be considerably greater than the amount required for air-conditioning.

Air-conditioning equipment may, of course, be transferred from one part of the mine to another, or, if necessary, from one mine to another. Arrangements for obtaining maximum benefit from the air circulation system cannot be made with equal ease when mining operations are discontinued in one part of the mine and are initiated on some new development. Also, when the orebodies are worked out, the shafts have no recoverable value, whereas air-conditioning equipment may be salvaged and used elsewhere.

The mechanical equipment is of very simple character, and involves no considerable increase in operating personnel. The underground plants require only the periodic inspection now given to booster fans by the shift bosses on their rounds, repairs being made by the regular force of mechanics.

Since the air entering such a conditioning plant circulates in the direction opposite to that of the brine, the warm air makes its first indirect contact with the brine about to leave the plant, whereby the final temperature of the brine is increased. So long as the initial temperature of the brine remains unchanged, and assuming correct design, there should be no change in the final temperature of the air. No matter how high the ground temperature may rise with increase of depth, it would therefore seem possible to cope with the growing amount of heat developed in the mine workings and to maintain a good standard of comfort in working conditions.

DISCUSSION

(*Lucien Eaton presiding*)

L. Eaton,* New York, N. Y.—The application of this system is especially good at Butte, because the shafts there are deep and small. It is a physical impossibility to hold a large shaft, and it is a physical impossibility to force enough air from the surface through small shafts down to their workings and cool them off. This new scheme solves both problems. We all know that the amount of air required for breathing underground is comparatively small; 50 cu. ft. per man per minute is a great plenty. If there is not outside oxidation, we can get along with considerably less, and yet the amount of air passed through the workings of some metal mines primarily for cooling and dedusting is as high as 800 cu. ft. per man per minute. The possibilities of power saving, as well as construction costs, are stupendous.

G. S. Rice,† Washington, D. C.—I am interested in this project of cooling mine air by a method which, so far as I know, is the first application in mining. Was the St. John del Rey plan considered in comparison? Did the brine lead to much corrosion of the pipes? Ceramic-lined pipes have been used in some mines in Germany for hydraulic sand filling and such lining might be a remedy for corrosion by brine.

Also how about siliceous dust settlement?

P. B. Bucky,‡ New York, N. Y.—How was the air circulated through cooling towers? Might it not be more efficient to operate the fans blowing, coursing their air supply through the cooling towers?

A. S. Richardson (written discussion).—In reference to Mr. Rice's desire for a comparison of the methods described in this paper with those used at the St. John del Rey mine in Brazil, it seems necessary to say that on account of the difference in climatic conditions the process used at Butte could not be used at the latter property. The first refrigeration plant installed at the St. John del Rey mine was a surface plant designed to cool and dehumidify the air entering the downcast shaft. Its operation has been fully described elsewhere. A later installation of two 75-ton Carrier units was made underground, but the writers have seen no published account of the results obtained.

In the surface cooling plant at Butte we did not use a spray pond. It was found that by forcing the brine, or water, to be cooled through spray nozzles, with fine atomization and a sufficient height of fall, better results could be obtained at less cost than would have been possible with an ordinary cooling tower. Water from the spray nozzles fell on to an inclined platform and thence flowed through a launder to a centrifugal pump, which again forced it through another set of sprays; in all, three sets of sprays were used.

The estimates as to the results that may be obtained from an underground air-conditioning plant were based on the operation of a surface cooling plant which during the summer months would be capable of cooling the brine to a temperature about midway between the atmospheric wet-bulb and dew-point temperatures. This process was still in the preliminary experimental stage when the paper was presented, but since then we have cooled water from 75° F. down to 48° F. with air having a temperature of 90° F., wet-bulb 60° F., and dew point 39° F. A number of details of construction have not yet been fully worked out. This part of the process is of little value in damp climates, but would make it effective in hot, dry, climates where otherwise it would not be applicable.

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‡ School of Mines, Columbia University.

Ventilation and Safety Practices at the Frood Mine of the International Nickel Company of Canada, Limited

By Ralph D. Parker,* Copper Cliff, Ont.

(New York Meeting, February, 1934)

The Frood mine (Fig. 1) is 2½ miles north of Sudbury, at a general elevation of 1000 ft. above sea level. It includes the original Frood location, which was the No. 3 mine of the Canadian Copper Co., and the adjoining Frood Extension, which until 1929 belonged to the Mond Nickel Co.

Mine Openings

The mine is operated from two vertical shafts and one underground incline shaft. No. 3, for men, supplies and ore, is a vertical six-compartment shaft in the footwall of the south mining area and extends from surface to a depth of 3045 ft. No. 4 is a vertical three-compartment shaft, 2500 ft. north of No. 3 and adjacent to the north mining area. It extends from surface to a depth of 3345 ft., and is used exclusively for hoisting ore. No. 5 shaft, inclined at 65° to the horizontal and of three compartments, is near No. 4 and extends from the 2600-ft. level to 77 ft. below the 3100 level. It is used for development below the 2800 level. The ventilation shaft, No. 1, with which this paper is directly concerned, will be described fully in the following pages.

Mining Methods

The orebody is laid out in 80-ft. sections at right angles to the general strike and numerically indicated from the southern extremity toward the north. Each section consists of a 45-ft. stope and a 35-ft. pillar, extending the full width of the orebody. Pillars and stopes are superimposed from level to level. The present mining territory is divided into two vertical areas, a north and a south area. Both areas have the same number of stopes on each level, the two areas being separated by 10 sections to be mined later. In a few instances, at the ends of the south area, where the orebody has narrowed, longitudinal stopes are used, of more than the usual 45-ft. width. It will thus be seen that mining operations have been concentrated in two relatively small areas, offering

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* Mine Superintendent, International Nickel Co. of Canada, Limited.

an ideal opportunity for effective ventilation with minimum resistance to air flow.

The mining methods depend upon the characteristics of the orebody in the sections being mined. Most of the ore is at present mined by square set-and-fill, the remainder by horizontal cut-and-fill.

VENTILATION

Prior to the adoption of the present mining system, it was foreseen that effective mechanical ventilation would be necessary. Large daily tonnages of high-sulfide ore were to be mined from wide stopes, involving the use of considerable timber. Conditions were present to cause high temperatures, high relative humidities, unhealthy dust accumulations, and possible accumulations of poisonous gases such as sulfur dioxide and hydrogen sulfide. Mine fires also were to be expected, as electrical apparatus and other possible causes were present. Although ventilation will not always prevent a fire, complete control of the air currents is indispensable, both to safeguard human lives and to aid in extinguishing the fire. Silicosis, while not common in the district, was regarded as a possible result of poor ventilation. The efficiency of the workers would be lowered from 10 to 50 per cent because the temperatures and relative humidities of the working places exceeded ideal working conditions. Injury frequency was also regarded as closely related to good working temperatures and humidities. Therefore, to obtain definite control of air current for fire protection, freedom from harmful accumulations of gas

FIG. 1.—SURFACE PLANT AT FROOD MINE.

and dust, low injury frequency, and the least decay of timber, the present ventilation system was planned.

Source of Heat

The chief sources of heat in the Frood mine are oxidation of sulfides in the ore and waste fill, oxidation in the decay of mine timber, and heat generated by underground electrical equipment.

The heat produced by the oxidation of sulfide minerals is the most important cause of the increased temperatures. The average sulfur content of ore being mined at present is 18 per cent. The oxidation in the decay of mine timber is a fairly important source of heat. The square-set-and-fill system of stoping necessitates the use of much timber, all of which must be left in the waste filling. The horizontal cut-and-fill system, because of heavy ground conditions, requires an unusual amount of timber, some of which is also left in the fill. Electrical equipment in this mine is responsible for a limited amount of heating. Pump motors, hoist motors, motor-generator sets, haulage motors, transformers, charging stations, and lighting generate sufficient heat to cause a temperature increase of 13° in 100,000 cu. ft. of air per minute. This heat is not directly noticeable, as the rock temperature tends to offset it. Most of this equipment is remote from the working places. The heat caused by friction of air currents against the sides of the airways is not an important source of heat, because the airways are straight, large in section, unobstructed, and comparatively short.

Surface and Rock Temperatures

The surface air conditions of this district are extremely favorable for efficient ventilation, as will be seen from the mean average, maximum and minimum temperatures of a period of 16 years given in Table 1.

TABLE 1.—*Temperatures for Sixteen Years*

Month	Maximum, Deg. F.	Minimum, Deg. F.	Mean Average, Deg. F.	Month	Maximum, Deg. F.	Minimum, Deg. F.	Mean Average, Deg. F.
January	18.3	0.7	8.8	July	76.3	56.0	66.2
February	20.5	0.7	10.5	August	74.1	53.3	63.7
March	33.2	13.4	23.5	September	65.4	46.4	55.9
April	47.7	28.4	38.0	October	52.5	36.2	44.3
May	60.6	39.6	50.1	November	37.4	24.4	30.9
June	72.3	50.8	61.6	December	25.0	9.5	17.2
							39.2

The approximate average relative humidity of the surface air in the summer months is 67 per cent. The precise figures of the relative humidity of the air in the winter months are not complete, but, from the available information, that of the entire 12 months has been calculated to be 75 per cent. From these figures, the true significance of the favorable air conditions in the ventilation of the mine may not be realized at first glance. However, the low moisture content of the surface air is very desirable, because of the high evaporative effect when raised to higher temperatures within the working places of the mine.

TABLE 2.—*Rock Temperatures*

Level	Temperature, Deg. F.	Level	Temperature, Deg. F.	Level	Temperature, Deg. F.
400	46	1600	55	2600	62.25
700	48	1700	55.75	2800	63.75
1000	50.5	2000	58	2950	65
1200	52	2200	59.5	3100	66
1400	53.5	2400	60.75		

The rock temperatures (Table 2) are ideal for ventilation conditions in that the highest temperature is below that of the ideal temperature desired; namely, 66° to 72° F. Another significant feature, in the relation of the rock temperature to the ventilation system, is its effect on the intake air in the cold winter months, when temperatures below zero Fahrenheit are registered. Warming of air to moderate temperatures to prevent water lines and air lines from being frozen is essential. As the areas being mined at present are below the 2000-ft. level, the intake air is confined to No. 1 shaft as far as the 2200 level. The interchange of heat between the rock and the air, and to a lesser extent the effect of the semi-adiabatic compression of the air in its downward course, are responsible for the increase in the temperature of the intake air, as illustrated in Table 3. The increase in temperature due to the effect of semi-adiabatic compression is considered to be not more than 2.8° F. per 1000 ft. of depth.[1] This effect therefore causes a temperature increase of approximately 7.5° F. between the surface and 3100-ft. level. The most important factor is the heat interchange between the downcast air and the shaft walls. For nine months of the year, as the mean surface temperature of the intake air is lower than the mean rock temperature, the rock in the vicinity of the downcast shaft is gradually cooled. Rock temperatures taken at a depth of 60 ft. in diamond-drill holes near No. 1 shaft showed that the rock temperature had fallen 4° F. in a period of 18 months. Ultimately the area surrounding the downcast shaft would assume a temperature slightly higher than the mean annual temperature of the surface air, which is 39.2° F. for the Sudbury district.

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[1] J. W. Walker and J. E. Jensen: Mine Ventilation and the Cooling of Air. South African Inst. Engrs. (Oct., 1929).

The crust of rock forming the periphery of the downcast shaft changes in temperature from season to season, and it is the temperature of this comparatively narrow zone that controls the temperature of the downcast air. During the hot summer months the crust is heated to a temperature of approximately 70° F. During a short spell of cool weather the intake temperature may drop to 45° F., but the temperature of the air drawn from No. 1 shaft below the 2200-ft. level is not appreciably reduced. In the early summer the intake air will be cooled as much as 5° F. in its path down the shaft. As a general rule the air is warmed in its downward path. Over a long period the cold air would also have a cooling effect on the temperature of the rock immediately adjoining the airway.[2] The deciding factor on this interchange of heat is the interval of time of contact. At higher velocities, the exchange of heat is less, and greater with lower velocities. In very cold weather, the quantities of fresh air delivered to the mine are regulated so that the velocities are low enough to deliver air to the working levels at temperatures above freezing.

TABLE 3.—*Increase of Temperature of Intake Air with Depth, No. 1 Shaft*

Level at Which Temperature Was Taken[a]	Velocities			
	2264 Ft. per Min.		1300 Ft. per Min.	
Surface	−5° F.	+35° F.	−5° F.	−50° F.
1200	27.5°	41.5°	38.5°	
1600	33.0	43.0	41.0	
2000	37.5	44.5	43.0	
2200	41.0	45.5	44.5	
2400	43.0	46.5	45.5	+12.5° F.
2600	44.0	47.0	47.5	
2800	45.0	48.0	48.0	+16° F.
3100				+26° F.

[a] Temperatures taken at the shaft before air entered crosscuts to be distributed.

Fan Equipment

The mine is mechanically ventilated by two pressure fans installed on the surface at the top of No. 1 shaft.

The fans are duplicates, being single-inlet, single-width, and of the superturbo type with backward-curved blades and inlet guards. The fan wheels are 10 ft. in diameter and are composed of 24 double-curved blades, of No. 8 gage material, mounted between a flange and a back plate, both of ⅜-in. machinery steel; this construction, with the double-curved blade, giving the greatest rigidity for a given weight. The wheels are mounted on a forged-steel shaft, 6 in. in diameter in the center and

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[2] Provision has been made in the fan housing for the installation of heaters at a later date should this be necessary.

5 in. at the bearings. The bearings are of the roller type and are supported on long structural-steel pedestals between the fans. This arrangement permits the removal of the fan wheel by way of the intake, without removal of the flexible coupling or moving the motor.

The fan housings are of ¼-in. sheet steel, reinforced with 3 by 3 by ⅜-in. angle bracing on 2½-ft. centers.

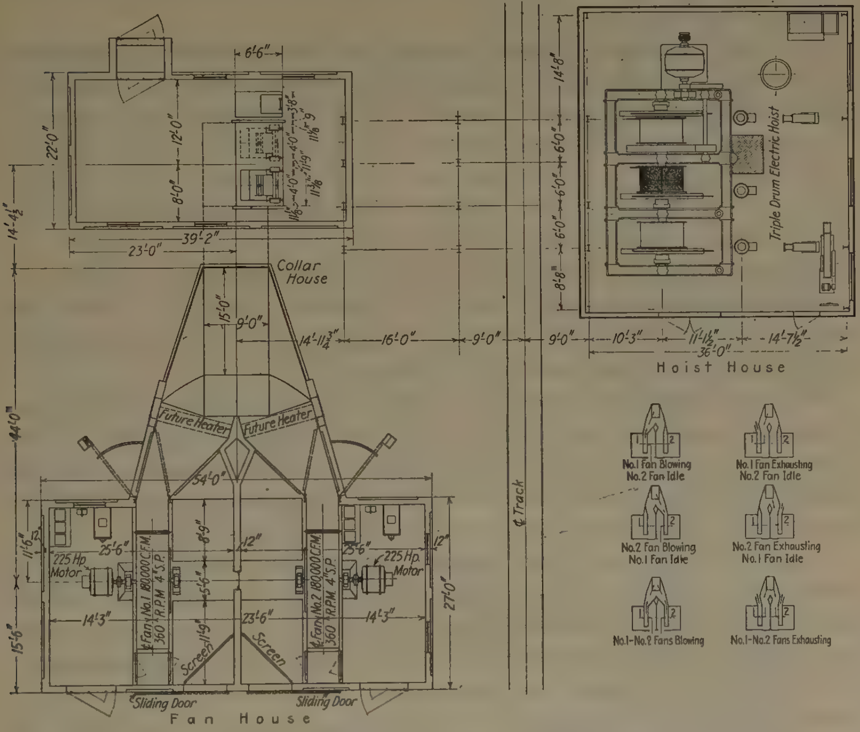


FIG. 2.—PLAN OF SURFACE EQUIPMENT AT VENTILATION SHAFT.

Each fan is capable of handling 180,000 cu. ft. of air per minute against a 4-in. static pressure when running at 360 r.p.m., requiring 190 brake-horsepower.

Each fan is direct connected to a separate 225-hp., 2200-volt, 40° C., three-phase, 25-cycle, 375-r.p.m. variable-speed, wound rotor motor. Each motor bearing is equipped with antifriction bearings, the slip rings are totally incased by a cast-iron cover, and all windings are impregnated with moisture-proof and acid-proof paint.

A large bank of "grid-type resistors" forms a part of the installation with each motor, together with a nonreversing secondary type controller. Half speed and various degrees of increase up to full speed are accomplished through the resistors, which are of sufficient size and capacity to permit continuous operation at these speeds. The switching equipment for each unit comprises: 1 type BH three-pole, single-throw, 400-amp., 7500-volt, manually operated remote control, and an automatic circuit breaker, all mounted on a 1¼-in. pipe frame. Protection is provided by two 5-amp. transformer trip coils, inverse time limit attachment, and 110-volt under voltage release.

Energy is supplied to the fanhouse from No. 3 shaft transformer room through a suitable B-16 oil circuit breaker and a three-wire transmission line.

The two fans and motors are installed in parallel in fireproof housings with reversing arrangements, as shown in Fig. 2. The fan building is entirely of steel and concrete, and the hood over the shaft entrance is ¼-in. sheet steel, reinforced with 4 by 4 by ⅜-in. bracing on 3-ft. centers.

The airway between the fans and the shaft is an incline opening, concrete lined, sloping from the fan hood to and entering the shaft 30 ft. below the collar (Fig. 3).

The speed control mentioned was provided because the volume of air necessary to circulate would vary: (1) with the seasons, (2) with the tonnage mined, and (3) with the mine resistance, which would gradually change as the workings increased in number.

Fans of the backward-curved blade type are best suited to the conditions described above, and they were installed because of their nonoverloading characteristics and more constant volume at varying pressures. Such a fan adapts itself readily to future changes in pressure caused by the increased number of openings underground.

Later developments have proved that these changes are not so important as the great variation in pressure produced by natural ventilation. Calculations since the installation of the fans have shown that the natural ventilation pressure varies from an extreme in summer of 1.35 in. of water against the fans to 3 in. with the fans in the winter, when the fans are operating in the blowing position. Very often the volume produced by natural draft in the winter approaches that delivered by one of the fans at full speed. Obviously this makes it advantageous to have a nonoverloading fan. Measurements made under certain conditions and fan speed in summer show a delivery of 270,000 cu. ft. per min., while a corresponding volume in winter is 370,000. This natural draft is not the result of difference in elevation of the collars of the intake and return shafts, as they are at approximately the same elevation.

The most important cause of natural flow is the temperature of the rock surrounding the intake shaft. In winter, with the fans blowing, a large volume of rock forming the walls of the intake shaft is cooled many degrees below the natural rock temperature. The reverse is true of the return system. The temperature of the rock in each place tends to keep the air at the entering temperature. Thus a column of air of lower density is maintained in the return system. As already explained, the big temperature change takes place in the stopes. The governing feature in the volume and pressure of the natural draft is the temperature on the surface and subsequently the temperatures in the intake shaft. The variation in the upcast system is only 3° F. throughout the year, whereas the temperature of the downcast reaches 90° F. During the latter part of the hot season the downcast air column is considerably warmer than the upcast. The warm downcast column therefore offers a dead load resisting the fan pressure in the hot weather. The output of a fan with

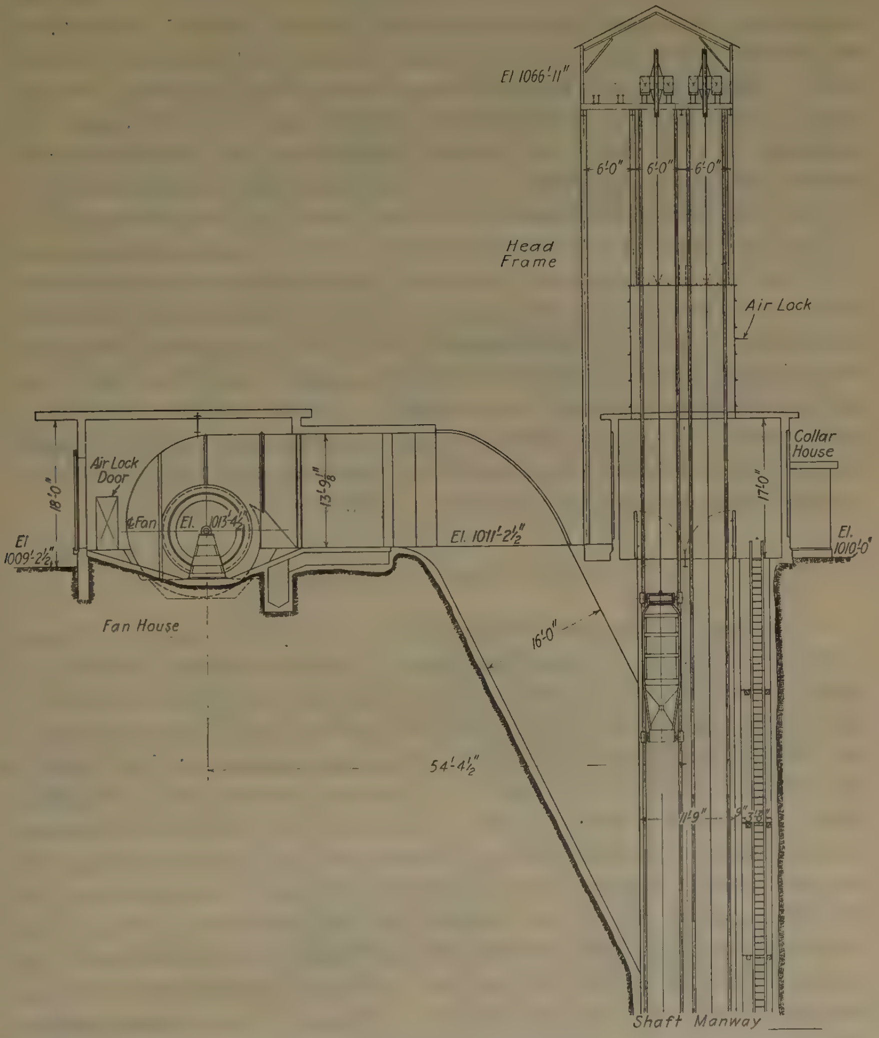


FIG. 3.—SECTION THROUGH FAN BUILDING, COLLARHOUSE AND VENTILATION SHAFT COLLAR.

forward-curved blade would fluctuate even more than the present fan with such a great natural pressure variation.

The fan with backward-curved blade, because of its flatter horsepower characteristic curve, is also better suited for motor drives since it allows the motor to be chosen at the rated conditions, with no fear of its being overloaded. As a general rule the motor for other types of fans is about 50 per cent above that actually required, in order to take care of overloads. This type has a lower efficiency rating than that with forward-curved blade but for the latter air dampers are required in the fan discharge to compensate for lowered resistances, and this appreciably reduces the mechanical efficiency. Thus, under the existing conditions, the fan with backward-curved blade gives better general efficiency.

The inlet guard is a special guard placed in the housing inlet, made up of several vanes so curved as to allow a free passage of air near the fan shaft, but, at the large diameter of the inlet, to turn the air in the direction of rotation of the wheel and permit it to enter at a velocity lower relatively to the wheel. This action produces forced vortex in the wheel inlet and reduces somewhat the capacity pressure and horsepower for a given speed. The shock losses are greatly reduced, so that the over-all efficiency is raised and the same air and pressure may be obtained at a slightly higher speed with considerable reduction in horsepower.

Ventilation Shaft and Equipment

No. 1 shaft, or the ventilation shaft, as it will hereinafter be called, serves two purposes. It serves primarily as an air course available for either blowing into or exhausting from the mine, and it also serves as an exit for men in case of emergency. It is inclined at 77° from surface to the 1200-ft. level and at 61° below this point to the bottom, which is 15 ft. below the 3100 level (Fig. 4). This shaft, throughout its length of 3388 ft., is lined with reinforced concrete and is completely fireproof. The roof support consists of 8-in. I-beams set in solid rock hitches 2 ft. deep and at 8-ft. 3-in. centers. These beams carry the reinforcing rods and the concrete, which was carried flush with the face of the beams. With the exception of the H-beams, the side walls and footwall are of the same construction as the hanging wall. Fig. 5 shows typical minimum cross-sections of the shaft at various horizons. The average effective area from surface to the 2800-ft. level is 159 sq. ft., and from the 2800 to 3100-ft. level, 112 square feet.

To equip the ventilating shaft as an emergency exit, double tracks of 60-lb. rail have been installed from surface to the 3100-ft. level, and specially designed man cages are maintained in readiness to hoist men in emergencies. A concrete-lined manway is also maintained from the 1200 level to surface (Fig. 3). Hinged grid-type safety platforms have

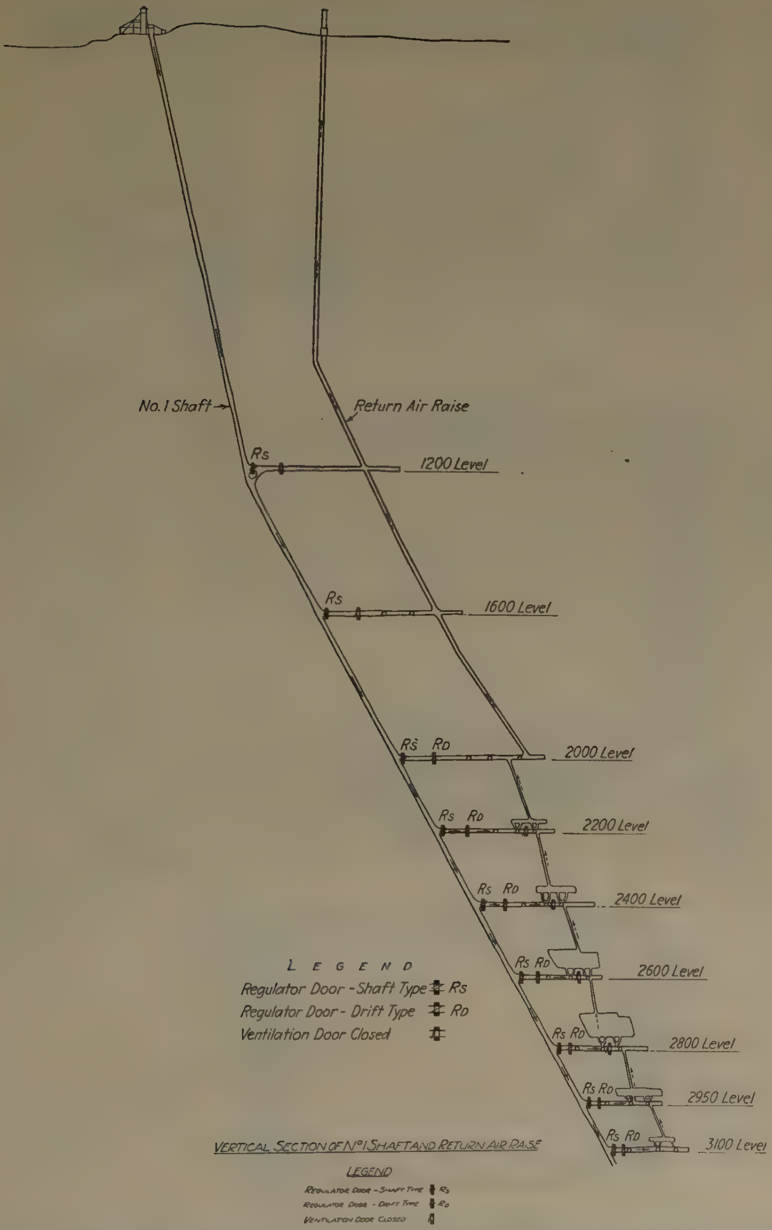
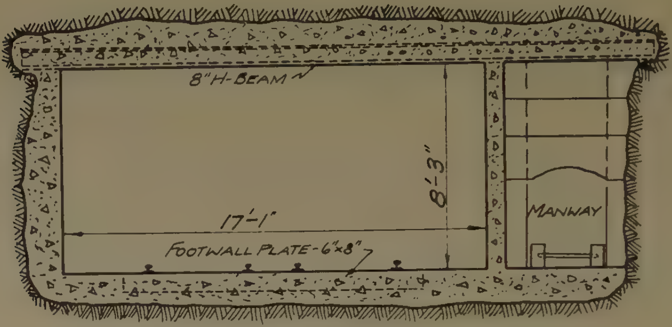
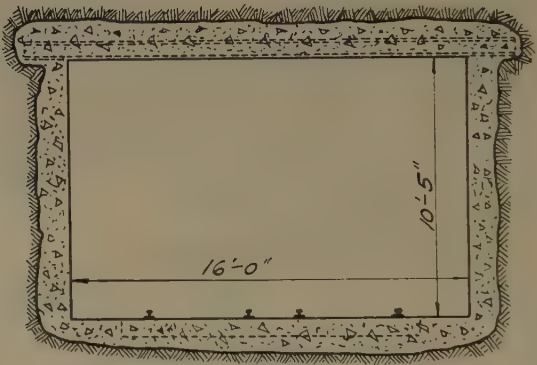


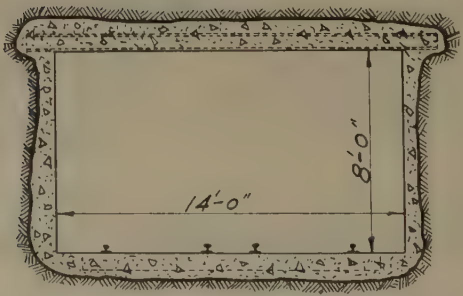
FIG. 4.—VERTICAL CROSS-SECTION THROUGH VENTILATION SHAFT AND RETURN-AIR RAISE.



SECTION SURFACE-1200



SECTION 1200-2800



SECTION 2800-3100

FIG. 5.—TYPICAL HORIZONTAL SECTIONS OF VENTILATION SHAFT.

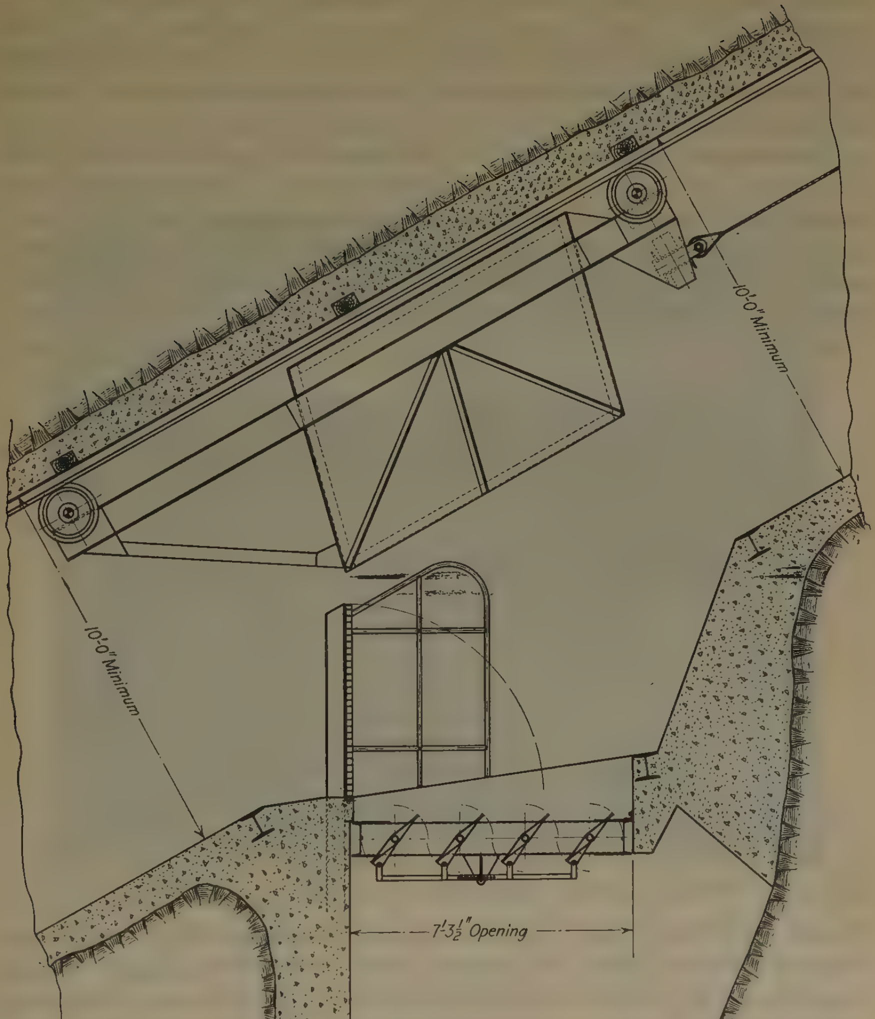


FIG. 6.—SECTION THROUGH VENTILATION SHAFT AT A STATION, SHOWING CAGE, GRID PLATFORM AND REGULATOR DOOR.

been installed at each level, to enable the men to enter the cages (Figs. 6 and 7).

The hoist signal system, installed at all stations, is similar to those in all other Frood shafts; namely, a three-wire common call and return call in each compartment.

A telephone system similar to, but independent of, the other mine systems has been made in the ventilation shaft. Each station has a telephone, as has the collarhouse at the ventilation shaft and the "toplander's" office at No. 3 shaft. All of these installations are on a three-wire common ringing system, and connections with other mine telephone systems are possible through the central switchboard at No. 3 shaft. Two 110-volt mains, the common wires for each compartment's signal system, shaft telephone lines, refuge-station telephone lines, and spare wires, are carried down the hanging wall of the shaft manway in one single-armored, lead-sheathed cable. Each wire in the cable is insulated with rubber and covered with a distinctively colored braid. The cable is 1$\frac{5}{16}$ in. outside diameter and contains 14 conductors.

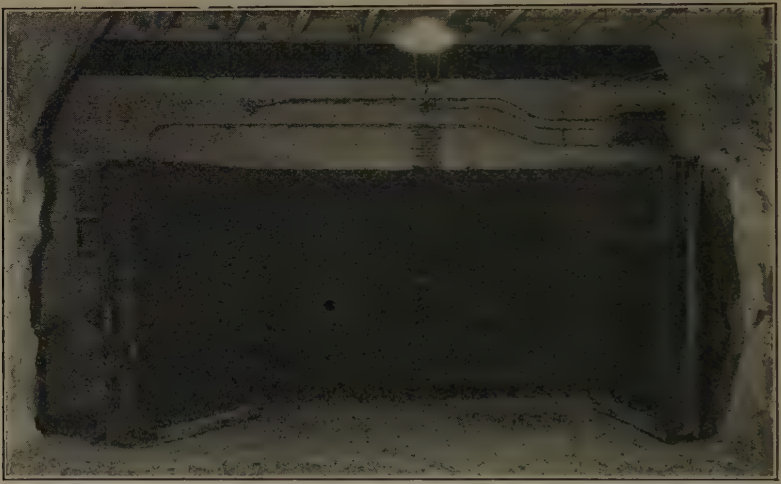


FIG. 7.—REGULATOR DOORS, OPEN, AND GRID PLATFORM AT VENTILATION SHAFT.

The collarhouse, entirely of reinforced concrete and steel, contains the emergency cages. When not in use, these are banked on top of heavy shaft doors in the shaft collar. The building is so constructed, however, that hoisting could be carried on simultaneously with the operation of the fans without affecting the ventilation. This is done by placing air locks at the entrances and a sliding bushing for the ropes in the air-lock doors over the shaft (Figs. 2 and 3).

The hoisthouse, of reinforced concrete, steel, brick, and tile, houses the electric hoist, hoist-switching and control apparatus, and a stand-by compressor.

All stations in the ventilating shaft are fireproof, as are all crosscuts leading from the shaft to the footwall longitudinal drifts. This was done by using concrete for walls and steel for back support. All doors, blocking on the back lagging, etc., are of steel. Figs. 7 and 8 show the construction and type of support used in the ventilation shaft stations and ventilation crosscuts. As described later, the level stations and crosscuts to the ventilation shaft are equipped as auxiliary refuge stations, in case of a fire in any part of the mine.

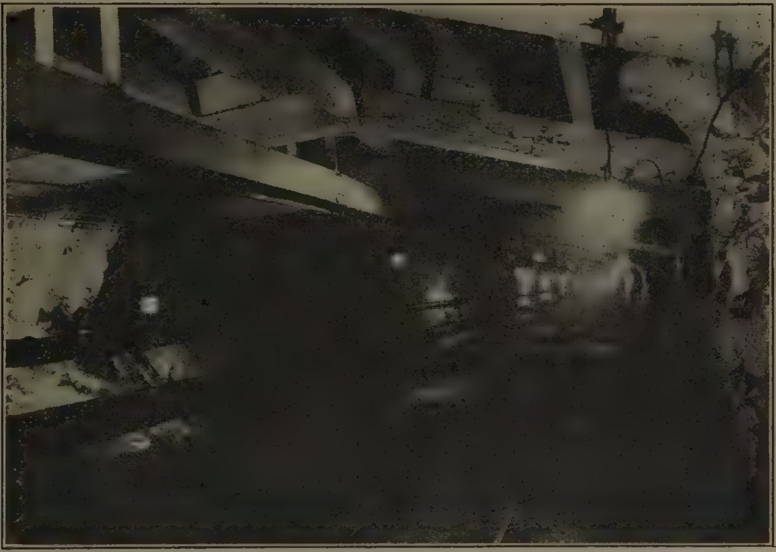


FIG. 8.—SADDLE-BACK METHOD OF TIMBERING MAIN CROSSCUTS.

As the housings of the hoist, fans, shaft collar and headframe, as well as the shaft, shaft stations, and crosscuts from the shaft, are fireproof, the ventilation and escapement way presents the minimum in risks from possible fires and the maximum safety and protection to the men.

Airways

Further evidence of the forethought exhibited in the planning of the Frood mine to assist ventilation is seen in the crosscuts, drifts, manways, raises, and all other airways. The crosscuts and drifts are of large area, straight, clean, unobstructed; and the roof support was designed to offer minimum resistance to air flow. In the main crosscuts (Fig. 8) the roof is supported by saddle-back timbering, the ends of the timbers being supported in hitches in the walls. In the other level airways the back of the drift is supported by an arrangement of steel I-beams adopted as standard for all drifts and crosscuts other than the main crosscut. This consists of placing 8-in. steel I-beams across the drift at 12-ft. intervals and 8 ft. above the top of the rail. These beams are supported by 2-in. rock pins set in the walls. Five-inch I-beams are placed on the 8-in. I-beams at 2-ft. intervals and blocked against the back of the drift. This fireproof construction offers little resistance to the flow of air, as there are no posts, spreaders, or other obstructions.

Main crosscuts were driven from No. 3 shaft on each level to intersect the orebody. On the main haulage levels these are 9 ft. high by 15 ft. wide; on the intermediate levels, 9 ft. high by 10 ft. wide.

Main footwall drifts were driven north and south from the main crosscuts, parallel to the orebody and approximately 65 ft. east of the footwall. On the main haulage levels, where trolley motor haulage is standard, this drift is 10 ft. high and 9 ft. wide; on intermediate levels, 9 by 9 ft. As this drift is a major airway, the point of minimum section being 8 by 9 ft. and its average section 8 by 10 ft., it is of great importance. (Typical level layout is shown in Fig. 13.) Longitudinal or extraction drifts, 8 by 8 ft., are driven in the orebody, parallel to its strike and at intervals of 44 feet.

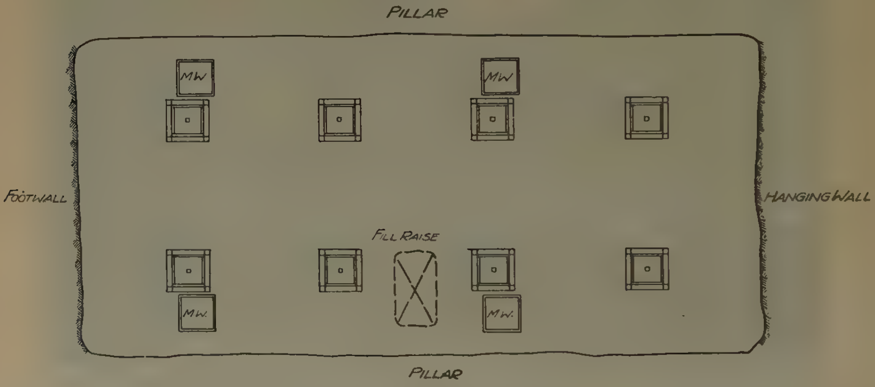


FIG. 9.—PLAN OF TYPICAL STOPE, SHOWING ARRANGEMENT OF CHUTES, MANWAYS AND FILL RAISE.

In the mining areas, crosscuts connecting the main footwall drift with the longitudinal drifts are driven at intervals of 3½ or 4½ stope sections. These crosscuts are 8 by 8 ft. and are unobstructed.

Boxholes are driven from the longitudinal drifts on 27½-ft. centers from pillar to pillar and 22-ft. centers from footwall to hanging wall (Fig. 9). A minimum amount of obstruction is offered in the longitudinals by the loading platforms used at all chutes (Fig. 10).

Stope manways are driven with boxholes and usually in pairs from the same longitudinal, but on opposite sides of the stope and adjacent to the pillars. Four manways or more are started with every stope (Fig. 9). Cribbed stope manways (Fig. 10) are 5 ft. 4 in. square, inside dimensions. All manways are equipped with a timber and steel chute

(20 in. high, 2 ft. wide, and smooth lined), a ladderway with platform at 14-ft. intervals, and air and water lines. At least 2 ft. clearance is maintained between all ladderways and platforms. This is necessary to permit the passage of rescue men in oxygen breathing apparatus without danger to equipment. In steeply inclined manways, ladders are offset at every landing. The minimum effective area available for air flow is 7.3 sq. ft., of which 3.3 sq. ft. is through the smooth-lined timber and steel chute.

One or two stope-fill raises are driven from every stope. In all cases, one fill raise is connected through to the hanging-wall longitudinal of the level above (Figs. 11 and 14). As discussed later, the connection of this fill raise is an important factor in controlling ventilation. Cribbed manways are maintained in these raises to provide exhaust-air outlets and auxiliary travel and supply ways. Stope-fill manways are essentially the same in size and equipment as the stope manways described previously. The platforms are latticed with 3 by 3-in. fir, while in the stope manways they are solid plank (Figs. 10 and 11). This construction of platforms offers the maximum effective area for air flow. Experiments showed that solid platforms reduced the volume of flow by 25 per cent. The minimum effective area for air flow in these manways is 12.4 sq. ft., as against 7.3 sq. ft. in the stope manways.

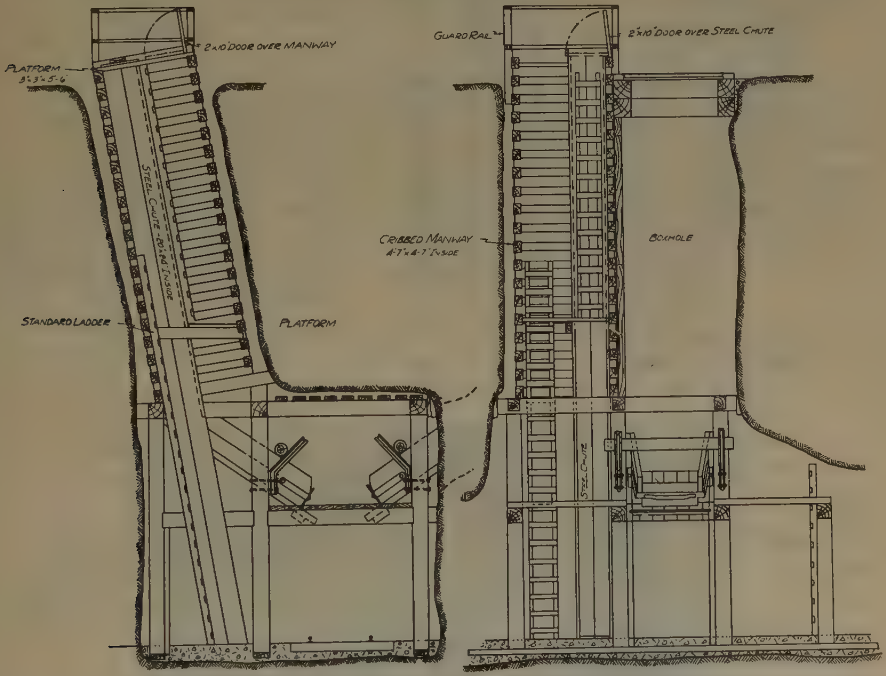


FIG. 10.—FROOD STANDARD COMBINED CHUTE AND MANWAY.

On the north side of each mining area, two or more stopes have been developed in advance of mining operations. Boxholes and manways have been driven to the stope floor and fill raises connected to the level above.

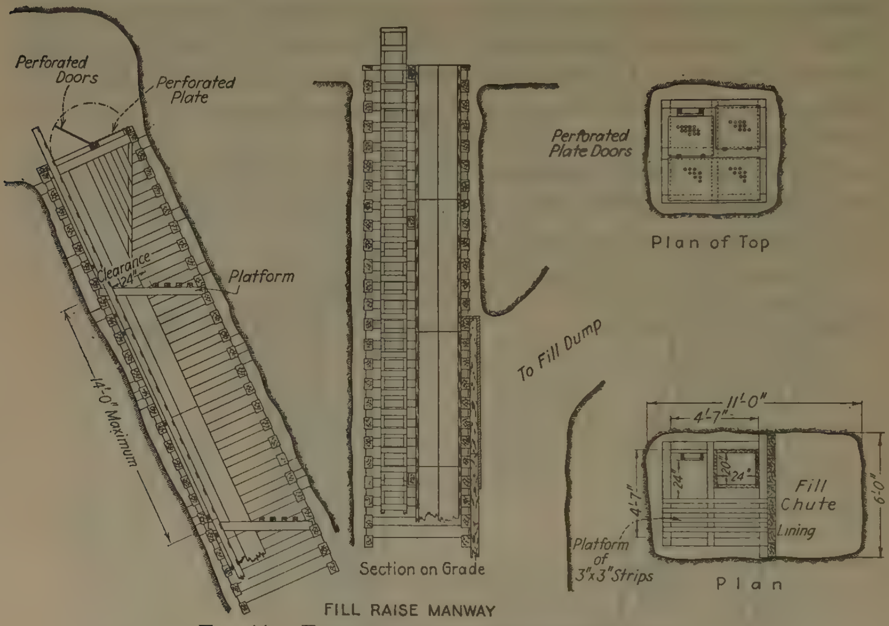


FIG. 11.—FROOD STANDARD FILL-RAISE MANWAY.

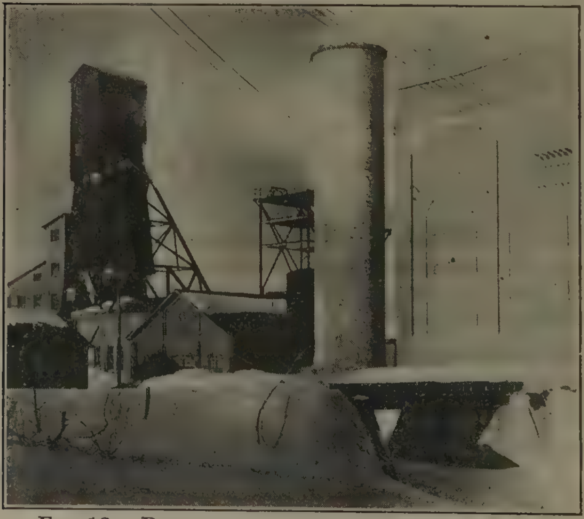


FIG. 12.—RETURN-AIR RAISE STACK EXHAUSTING.

Stope manways and boxholes from the return-air longitudinal (hanging wall) have been opened to provide a maximum opening for flow of air. Stope-fill raise manways are stripped of all platforms, ladderways, and steel chutes, and the fill chute emptied to allow a minimum sectional area of 64 sq. ft. in each raise. All boxhole chutes and manways leading to the stope floor from fresh-air longitudinals are kept closed.

The return-air raise (Fig. 4) extends from the 2000-ft. level to surface. The minimum dimensions of the different sections are:

	FEET
Surface to 200 level (vertical)	10 by 11
200 level to 400 level (sinuous)	10 by 11
400 level to 1000 level (vertical)	7 by 17
1000 level to 1400 level (63° inclination)	7 by 17
1400 level to 1700 level (62° inclination)	7 by 17
1700 level to 2000 level (57° inclination)	7 by 17

A reinforced concrete collar, resting on 18-in. steel bearers 22 ft. below surface and extending to 13 ft. above, forms a foundation for the stack (Fig. 12), which is built of hollow tile, 50 ft. high and 10 ft. inside diameter.

With a flow of 200,000 cu. ft. of air per min., the theoretical resistance of the intake shaft is 0.41 in. of water, of the mine proper 0.50 in., and of the return-air raise 1.0 in., or a total of 1.91 inches. The resistance factors used for the various airways are:

Intake shaft	0.0,000,000,015
Main footwall drifts and crosscuts	0.000,000,008
Longitudinal drifts	0.00,000,001
Fill raise manways	0.00,000,001
Return-air bypass	0.000,000,008
Return-air raise	0.000,000,006

Should future requirements demand more air, the surface fans are of sufficient size and capacity to deliver 400,000 cu. ft. against a resistance of 2.6 inches.

Air Distribution

With airways as described, the distribution of the air is not a complicated problem (Figs. 13, 14 and 15).

The fresh air enters the mine through the ventilating shaft and is supplied to each level through No. 1 shaft crosscut and regulator doors at the shaft (Fig. 7). The volumes distributed depend entirely upon the requirements of individual levels. Adjoining the main footwall drift, No. 1 shaft crosscut branches into a north and south split (Figs. 13 and 17). Regulator doors placed in these splits control the volume of air passing through to the main footwall drift and thence through the crosscuts to the north and south mining areas; the footwall longitudinals become the fresh airways from which the air passes through boxholes and manways to the stopes above. Should the orebody require more than two longitudinal drifts, two or more of these are fresh airways. The volumes delivered to individual stopes are controlled at the top of the fill-raise manways (Fig. 11). The vitiated air from each stope passes up through the stope fill-raise manways to the hanging-wall longitudinal drift of the level above. The advance stopes on the north end of all mining areas act as return-air bypasses from the hanging-wall longitudinal to the level above. Return air, therefore, emptying into the hanging-wall longitudinal, passes north and up through the return-air bypass raises to the waste-air gathering level—the 2000-ft. level (Fig. 16). The fresh-air longitudinals and return-air longitudinals in all areas are separated by doors so placed that the fresh air cannot be diluted by the return air.

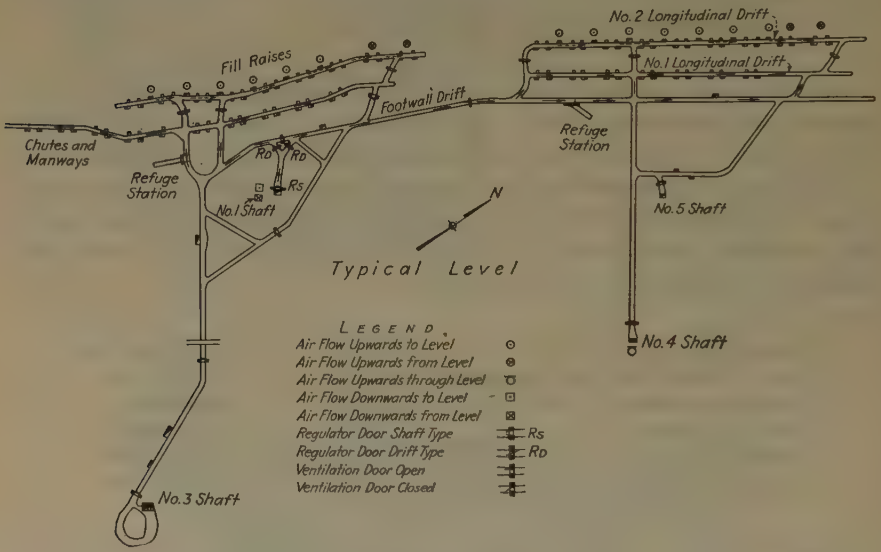


FIG. 13.—PLAN OF TYPICAL LEVEL.

On all levels, where connections have been made between the ventilation shaft and Nos. 3 and 4 shafts, small quantities of fresh air are permitted to pass into the crosscuts leading to these shafts. This serves to retard timber decay in the main crosscuts and keep all main passageways in fresh air. These accumulations of fresh air upcast the main shafts (Nos. 3 and 4) to the extent of 20,000 cu. ft. per min. each.

To prevent dilution of fresh air within the stopes, all chutes leading from active stopes to the hanging-wall drifts, in which the return air from the lower levels is gathered, are bratticed as soon as they fall into temporary disuse. The strips of brattice cloth are easily torn off when it is necessary to use the chute again. Footwall chutes are kept open as much as possible, even when in use, to allow free passage of fresh air to the stopes from the footwall longitudinals.

The 2000-ft. level acts as the return-air gathering level (Fig. 16). Both the main footwall drift and the Nos. 1 and 2 longitudinals drifts are available as airways to the return-air raise.

At this point several outstanding features of the ventilation system should be mentioned: (1) The operation of a horizontal split of the fresh air on each level; (2) the operation of a vertical split in each area, so that fresh air is not diluted with return vitiated air; (3) all important tramming and mining areas are maintained in fresh air; (4) return air is confined to the less congested hanging-wall longitudinals, or to raises and drift headings entirely outside the mining areas.

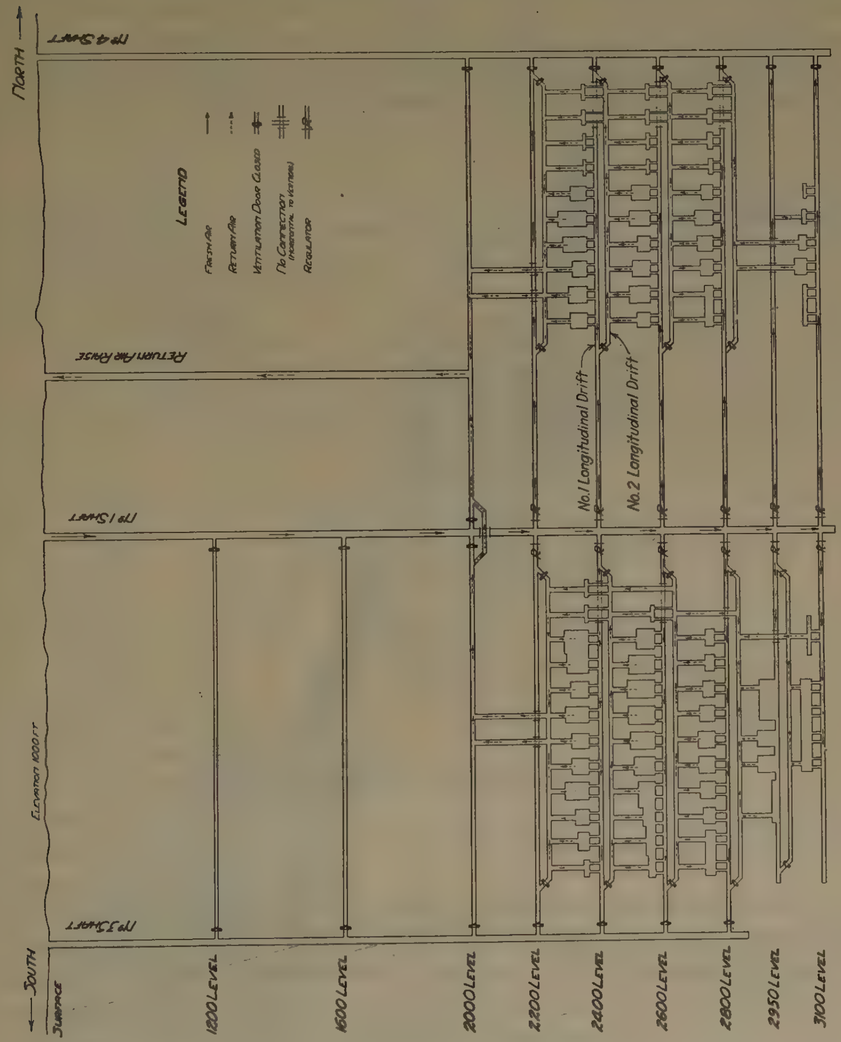


FIG. 14.—DIAGRAMMATIC VERTICAL SECTION OF VENTILATION SYSTEM AT FROOD MINE.

Air Regulation

Although the development was planned to minimize the number of doors and regulators, many have had to be placed to control the air currents (Figs. 13 and 16). Doors and door frames have been standardized, so that all of the same type are interchangeable and may be set on either side of the frame. All door frames are of whitewashed reinforced

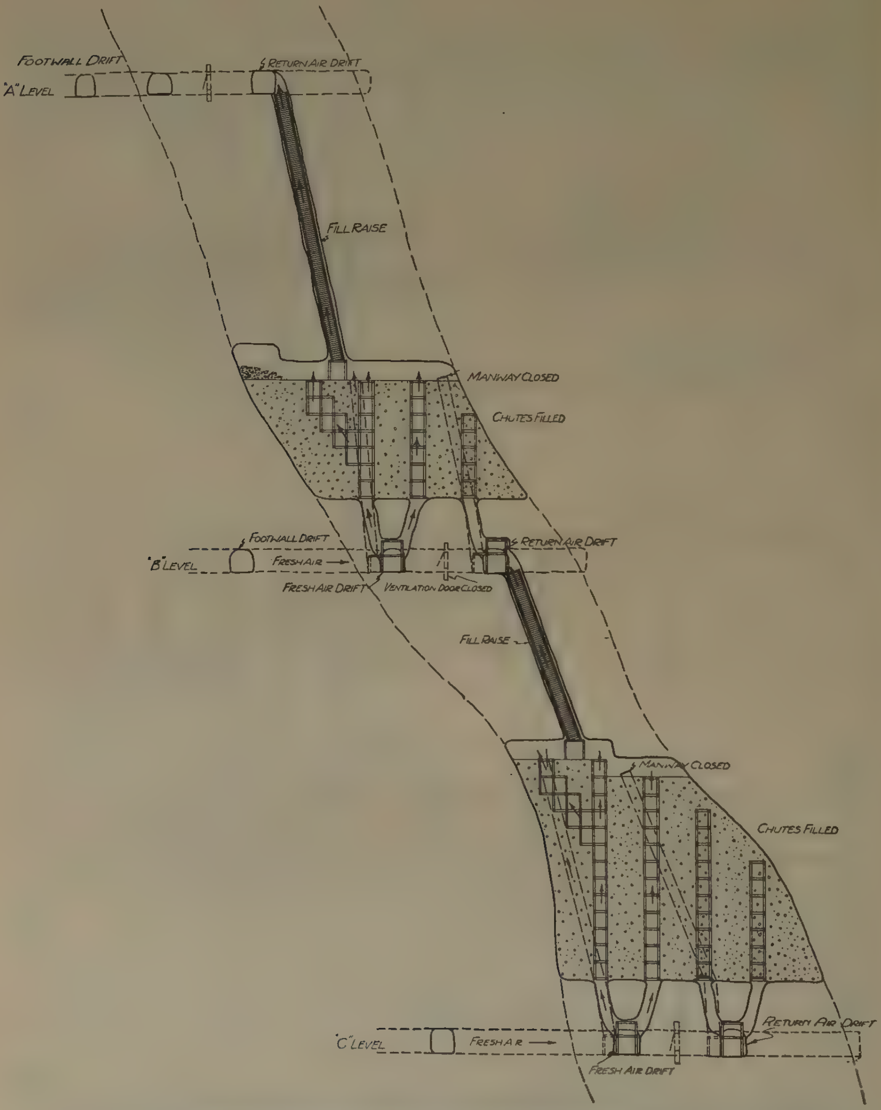


FIG. 15.—VERTICAL CROSS-SECTION THROUGH STOPES SHOWING AIR TRAVEL.

concrete and are secured to the bottom, walls, and roof by reinforcing rods (concrete hooks) set 2 ft. into the solid. Doors close flush against the frame. Sill openings are limited in size by the use of concrete sill blocks. Trolley-wire openings have a special "carry through" to prevent leakage. No openings are allowed around conduits and pipes. All ditches are enclosed boxes. Ventilation doors are painted black to make the unit conspicuous and so prevent injuries to motor-haulage crews. Red and green electric lights, controlled by the position of the door, are placed on both sides of mechanically controlled ventilation doors as an accident-prevention measure. All fire doors are painted red, for information in case of an emergency.

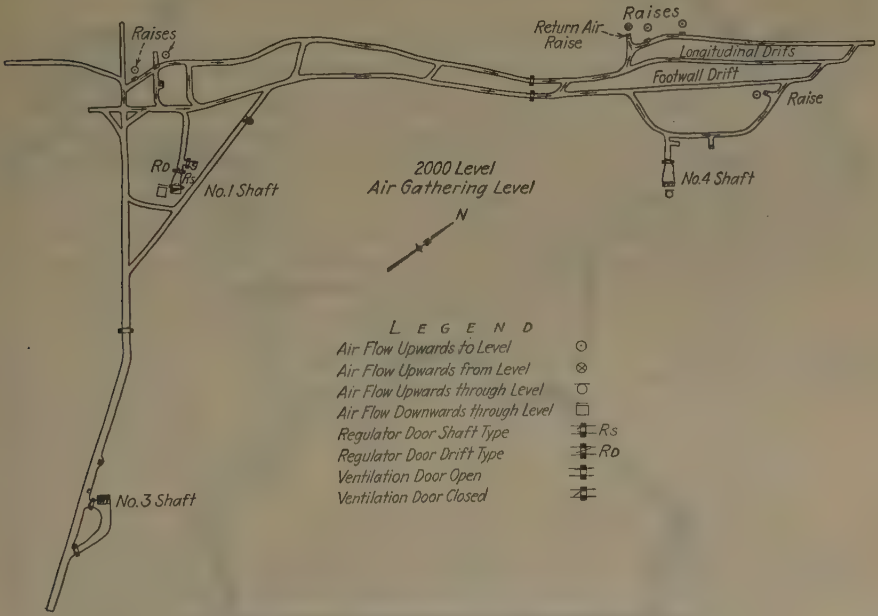


FIG. 16.—PLAN OF RETURN-AIR GATHERING LEVEL.

No. 1 Shaft Regulator Door (Figs. 6 and 7).—At the ventilation shaft on each level, fresh air enters through standard regulator doors of the "louvre" type. They are steel double doors, set in concrete frames with opening 7 ft. 3½ in. by 12 ft., and are hinged to open toward the level station. The louvres may be set to any opening by manipulation of one bolt in each door. If necessary, one or both doors may be fully opened. These doors offer the maximum flexibility in controlling the quantity of air delivered to any level.

The figures in Table 4 show that to maintain approximately the same splitting of the air in summer and in winter, wide adjustments to the regulator-door settings are necessary at each of the intake levels.

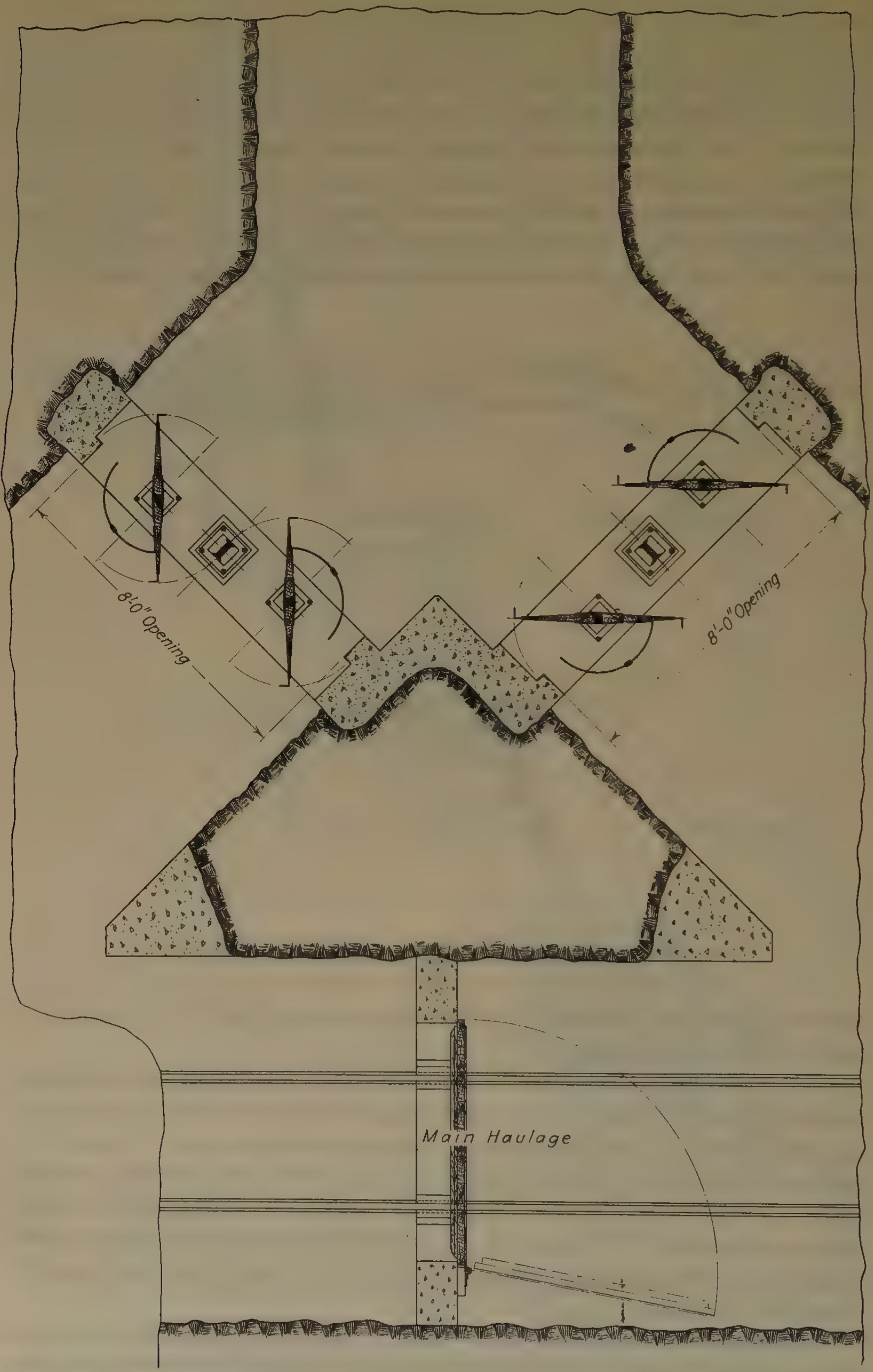


FIG. 17.—PLAN OF AIR SPLIT AND "BUTTERFLY" CONTROL DOORS IN VENTILATION-SHAFT CROSSCUT.

TABLE 4.—*Settings of Regulator Doors*

Level	Summer		Winter	
	Door Setting	Volume of Air, Cu. Ft. per Min.	Door Setting	Volume of Air, Cu. Ft. per Min.
2400	4	65,000	12 (open)	72,000
2600	4	46,000	5	52,000
2800	4	40,000	2	34,000
2950	4	29,000	1	16,000
3100	12 (open)	27,000	1	16,000

No. 1 Shaft Crosscut "Butterfly" Control Doors (*Fig.* 17).—The two splits of No. 1 shaft crosscut on every level have "butterfly" control doors; each set of doors being installed in a reinforced concrete frame with a total opening of 9 by 10 ft. There are two doors to each set, 4 ft. 2 in. by 8 ft. in size, each mounted on a vertical center shaft, which in turn rotates on roller bearings. The doors are of steel, with cover plates at top and bottom to prevent leakage. They operate horizontally and may be locked in any position desired by a clamping arrangement at the top of the door. The doors close against a central removable shaft and 3-in. shoulders on the sides of the frame. Because of the equalization of pressures effected by their design, these doors may be opened and closed easily, regardless of high pressures.

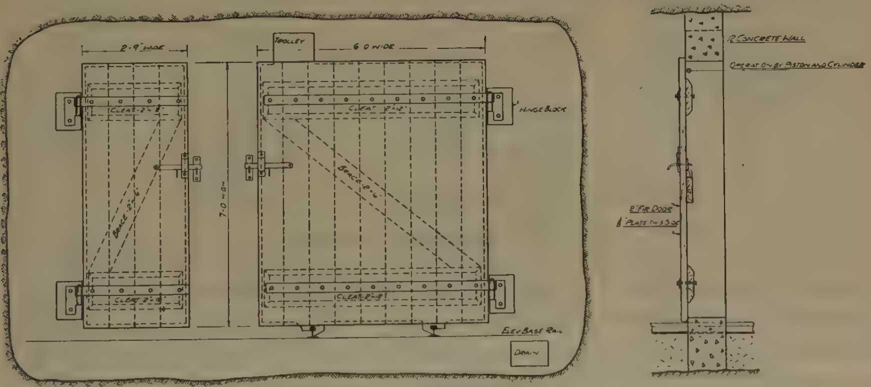


FIG. 18.—STANDARD VENTILATION DOOR.

Main-haulage Ventilation Doors (*Figs.* 18 *and* 20).—In all main haulageways, the standard ventilation-control installation consists of a single large motor-door and a small man-door, both in a common concrete frame. They are of 2-in. planks, diagonally braced. The front is covered with light plate and reinforced with a 3-in. angle-iron bumper. The large door is operated by a compressed-air cylinder and counterweight. The cylinder is controlled through a valve mechanically operated by wire ropes and levers placed 100 to 150 ft. on either side of the door (Fig. 20). Red and green electric lights, set on either side of these doors, are controlled by the door's position (showing red until it is wide open) and are used for the information and protection of haulage crews.

Ventilation doors in crosscuts, or elsewhere where transportation and travel is light, are large single doors in a concrete frame. They are similar in size and construction to the main-haulage ventilation doors but are operated by hand. Ventilation doors in timbered areas that may also be used for fire breaks are of 3⁄16-in. steel plate with angle-iron frame, reinforced with a diagonal cross brace. They are the same size as the other ventilation doors, are fitted with a pressure release slide, and have a standard latch and counterweight for automatic closing.

Fire Doors (*Fig.* 19).—Fire doors in the main crosscuts and at No. 3 shaft, where the transportation is heavy, are similar to the main haulageway doors in that they consist of a small man-door and a large motor-door set in a common concrete frame. They are the same size as haulageway doors, but are operated by hand. They are fireproof and automatically closed by counterweights. All fire doors are painted red. Two large tins of moist clay are kept near by for sealing openings in emergencies.

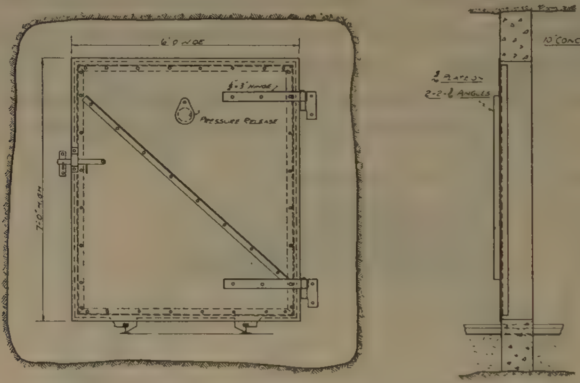
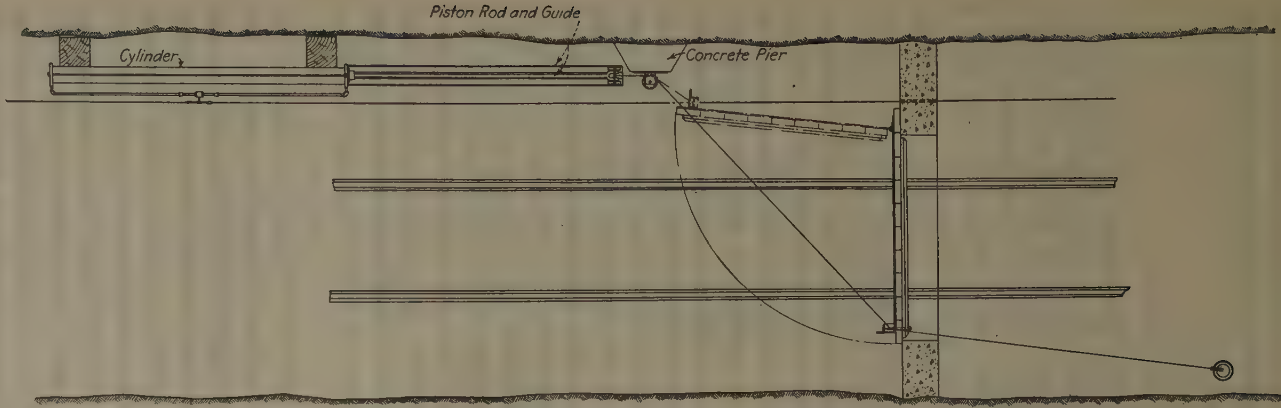
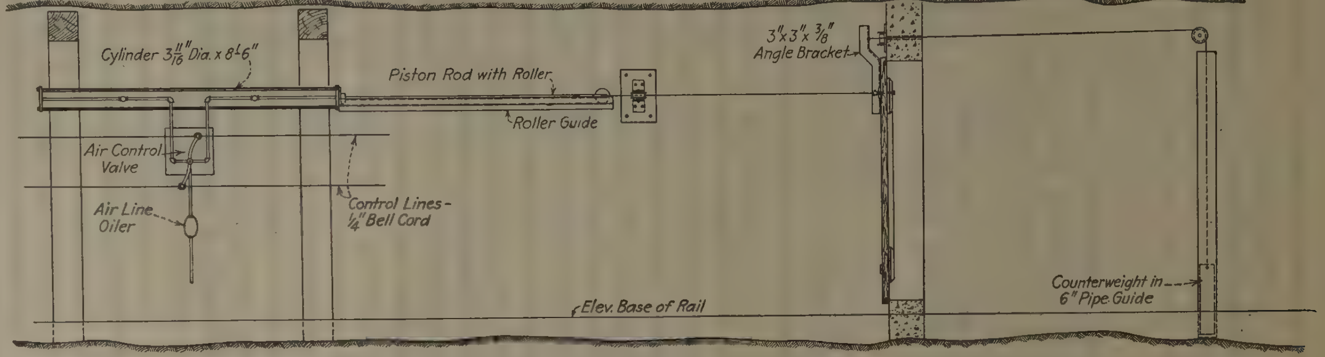


FIG. 19.—STANDARD FIRE DOOR.

Stope-fill Raise Manways (*Fig.* 11).—Fill-raise manways have hinged doors over the steel chute and ladderway openings. They are 2 by 2 ft. by 2 in. in size, constructed on a ¼ by 2-in. strap-iron frame covered with 3⁄16-in. perforated plate. The plate is perforated with 15⁄16-in. holes, giving an effective opening of 46 per cent for air flow. Doors are hung on heavy forged hinges and all have handles, available from either above or below. All stope-manway doors are equipped with a chain locking arrangement, for locking them from the inside or outside of the manway. The remainder of the top of the fill-raise manway is covered with perforated plate, like that used in the doors. This arrangement



Plan


Cylinder 3 11/16" Dia. x 8'-6"
Piston Rod with Roller
Roller Guide
Air Control Valve
Air Line Oiler
Control Lines - 1/4" Bell Cord
3"x3"x 3/8" Angle Bracket
Counterweight in 6" Pipe Guide
Elev. Base of Rail

Vertical Section

FIG. 20.—AIR CONTROL FOR MAIN-HAULAGE VENTILATION DOORS.

gives complete control of the airflow through the stope below. The effective opening of the top with both doors closed is 6.8 sq. ft. and with both open is 12.5 sq. ft. The latter figure can be increased to 21 sq. ft. in an emergency by removing all top plate.

Auxiliary Ventilation

Venturi injectors of 12-in. diameter and model 250 Vano blowers are used for ventilating dead ends. Standard practice has been to supply fresh air to all development headings whose faces are 200 ft. or more from fresh air, by means of auxiliary ventilation. The Venturi injector with a ¼-in. compressed-air nozzle is most often used, because its first cost is low, it is simple to install, and requires no maintenance.

Dust Control and Prevention

The danger to the health of employees caused by the liberation of dust into the air has been fully recognized by the operators of the Ontario mines, and every effort is being made to combat it. For gaging the relative effectiveness of the methods of controlling and preventing the concentration of fine dust in the air, and checking retrograde practices, a simple but reliable method of estimating the dust content of the mine air was necessary. Much valuable research work on the subject of dust estimation has been done in the mines of the Porcupine district, and the Frood mine has followed as closely as possible the procedure established in that district, in order that the figures for dust counts may be comparable with those of other mines of the province.

Dust Estimation.—At the Frood samples of the air are taken by the circular konimeter, followed by the use of a microscope having semi-dark ground illumination with a magnification of 150 diameters for counting the dust spots. By the circular konimeter, a modification of the original Kotze konimeter, 28 samples can be taken without changing the microscope slide in the instrument.

The method adopted at the Frood mine for treating the slides before counting on the microscope differs from standard practice, because the particles forming the spot on the slide consist almost entirely of soft sulfuric material rather than the hard silica particles observed in most mines. It is considered necessary to wash the slides to be examined, with hot distilled water and alcohol. After the slides are dry the circular slide is placed upon the Watson microscope stage for counting the number of dust particles in each sample. To facilitate counting, an ocular micrometer is placed in the eyepiece. The micrometer is ruled with two 18° sectors, and two parallel lines are etched on the micrometer so that they appear 5 microns apart. All particles larger than 5 microns in diameter are neglected. The total number of particles counted in both of the 18° sectors, multiplied by two, gives the number of particles per cubic centimeter of air.

The type of illumination plays an important part in observation of the number of particles. A modified form of dark background illumination is used. An Abbe NA 1.2 condenser, with a 9-mm. stop is used, and the source of light is a 100-watt, ½-watt, frosted-glass bulb, set 9 in. from the microscope and 4 in. from the table.

Dust-control Measures.—The ventilation system has been very effective in controlling the dust content of the mine air. Approximately 5000 cu. ft. per min. of dust-free air is diverted to every working place, so that the dust content is reduced by mechanical dilution to very low concentrations. The air current sweeps the dusty air from the stopes as fast as it is created by drilling and mucking operations. The system whereby return air from each level is immediately bypassed to the return-air system insures that the air exhausted from the stopes is confined to the hanging-wall longitudinals. The return bypass raises at the northern extremity of each mining area prevent the exhaust air from entering the stopes of the level above.

The ventilation, both primary and auxiliary, causes such a positive circulation of air that little smoke or fume can be seen or smelled one hour after blasting. The provision of the separate return-air raise, from the main gathering level to the surface, has insured that the men coming off shift after blasting are hoisted in a shaft free of smoke and dust.

In order to gain the fullest possible benefit from the ventilation system, it was necessary to arrange the times for blasting and for lowering and hoisting men in such a manner that the mine can be swept clear of dust and fumes before the men enter their working places. There are two working shifts per day and the blasting is confined to the lunch hour and the end of each shift. The blasting permitted at the beginning of the lunch hour is restricted to blockholing, and men do not return to the working places until half an hour after blasting. Most of the blasting is confined to the end of the shift, where a much longer interval is practised.

The system whereby the upper levels blast before the levels below them insures that the men are out of the working places before the ascending air currents bring up the smoke and dust of the blast from the level below. The men coming off shift walk in fresh air from the check-out boards to the No. 3 shaft stations.

The afternoon shift does not reach the working places until an hour and a half after the day shift has blasted.

Dust Prevention.—In addition to the use of the ventilating currents to dilute and eliminate dust, every effort is made to prevent dust from being liberated into the mine atmosphere. Wet drilling is practised in all working places. All advancing drifts and crosscuts are sprayed with water during blasting. This is done by connecting the air and water lines by means of (or through) a short piece of ½-in. hose, which acts as a crossover for the water into the air line. The water blast is turned on after the fuses are lighted, and allowed to operate for at least 15 minutes.

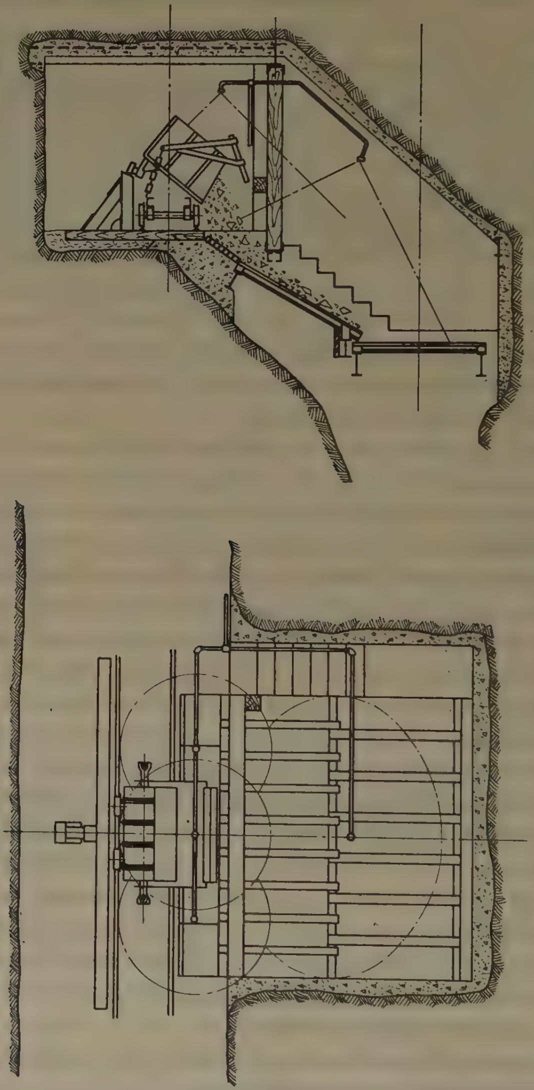


Fig. 21.—Water-spray installation at ore-pass grizzly.

All stopes are provided with water blasts, to wet the muck piles and timber in the immediate vicinity of a blast. This is done before the work of drilling, timbering or mucking is started. Water sprays (Fig. 21) are installed at all ore and waste-pass grizzlies. They are designed to interpose a curtain of spray between the falling rock and the drift. The ore-pass transfer chutes and the crusher on the 2800-ft. level are equipped with sprays to prevent the liberation of dust. The measuring bins at the loading pockets of shafts 3 and 4 have sprays to settle dust during loading operations.

FIRE PREVENTION

All the buildings constituting the surface plant are of steel and brick, with Barrett asphalt roofing, and are fireproof so far as it is possible to build.

Standard fire hydrants, with a water pressure of 74 lb., are in the yards, two hydrants for fighting fire being available on each side of any building.

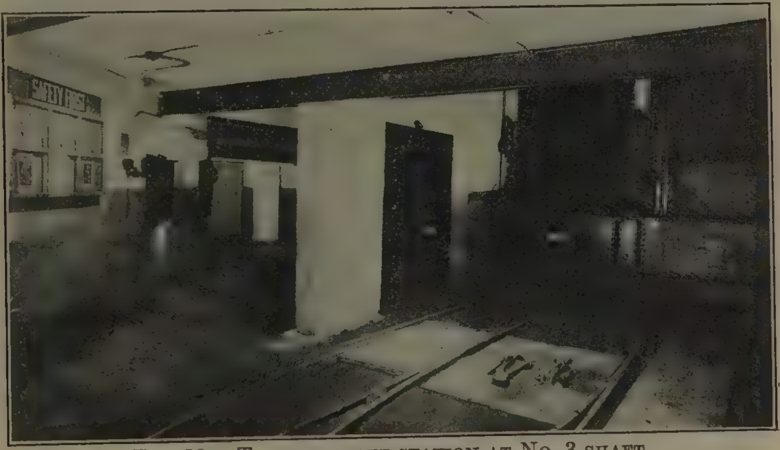


FIG. 22.—TYPICAL SHAFT STATION AT NO. 3 SHAFT.

Fire lines, of 2-in. pipe with outlets placed at convenient points, are installed in all buildings. Fire-hose reels, containing 100 ft. of 2-in. fire hose fitted with ¾-in. nozzle, are hung on the wall at each outlet.

Fire extinguishers for immediate use are maintained in all buildings. The types in use are the Fyre Fyters, Pyrene and Guardene. Fire guns, loaded with fire-extinguishing liquid at 100 lb. pressure, are maintained in the electrical shop and power house where fire resulting from high-voltage current may occur.

There are four hose houses at vantage points in the yard, each containing fire hose mounted on hose-reel carts. In each of three hose houses, 500 ft. of 2½-in. hose is available, while in the one at the extreme north end of the timber yard there are four fire reels, each carrying 500 ft. of 2½-in. hose. Each reel is mounted on 3-ft. gage rail trucks, which can be switched to the various hydrants in the timber yard or be run on the cage and lowered to any level in the mine where fire hose may be required.

The fire brigade is composed of members of the mechanical force, the Master Mechanic being in charge of the brigade and of all fire apparatus on the surface. Fire practice is held once each month. A fire-signal code is posted in the compressor room, all shops, and in the shaft house. Should a fire occur, the compressor man is notified immediately and it is his duty to sound the fire siren, which is mounted on the roof of the power house, and to indicate the location of the fire. By referring to the signal code, the proper number of blasts to be blown for any particular place can be learned.

Shaft stations at No. 1, No. 3, No. 4 and No. 5 shafts are built of reinforced steel and concrete. Both walls and back are thus reinforced for a distance of 20 to 50 ft. from the shaft. All shaft stations are closed off from the main crosscuts by standard steel fire doors (Fig. 22).

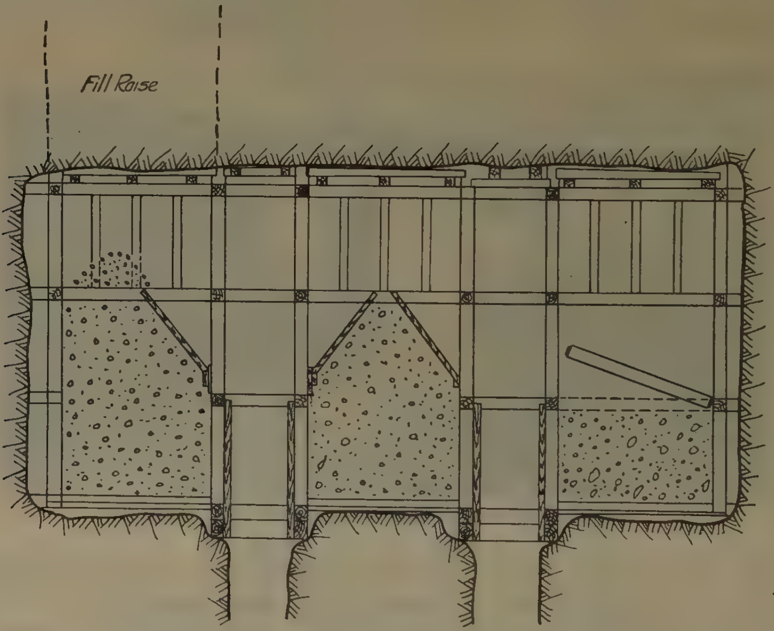


Fig. 23.—Sketch showing long girt application and removal.

The timbering in the main crosscuts conforms with the standard method of steel timbering for main crosscuts, detail of which is given under the heading of Ventilation. Smoking is prohibited in stopes, manways, cages or skips. Refuse cans are placed at convenient places throughout the mine. These are emptied each day and their contents sent to the surface for disposal.

All broken timber is collected at the manways in the stopes on day shift and is removed by nipping crews on the night shift. If it is necessary to keep an amount of scrap timber underground, this condition is set forth in a weekly refuse report from the foreman to the Safety Department.

When a stope has been mined to a desired elevation, all square-set timber, except one row of sets running from footwall to hanging wall along the north and south pillars of the stopes, is removed. This space is then back-filled with a fill of finer mesh than is used for ordinary filling. Thus a complete horizontal fire break is effected at the top of each stope.

A vertical fire break has been made at No. 20 section in the South mine, extending from the 2800 to the 2200 level, by use of cut-and-fill mining, so far as possible, in this area. When ground conditions make timber support necessary for safe mining, a modified square set is used, which is 5½ by 11 ft. instead of the 5½ by 5½-ft. standard set. The long measure of this set runs girtwise, or at right angles to the pillars of the stopes. As the filling operations advance, the long girts are removed, and a vertical fire break is maintained (Fig. 23).

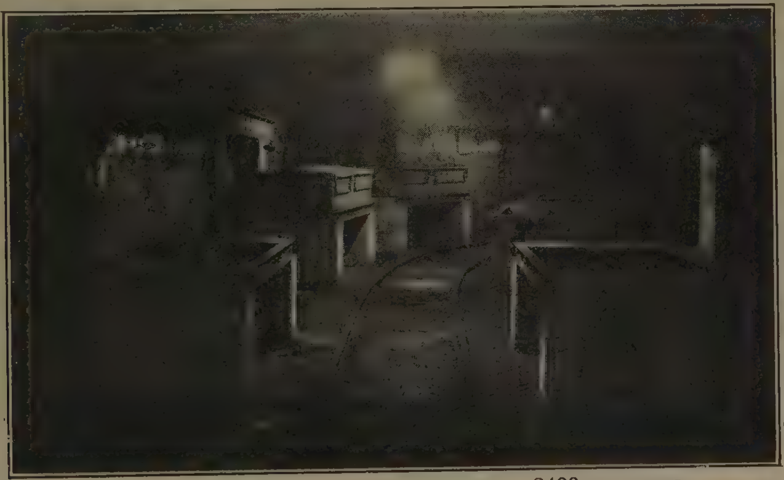


FIG. 24.—BATTERY-CHARGING STATION ON 2400-FT. LEVEL.

All electrical equipment underground is installed in fireproof stations of reinforced concrete and steel, with standard steel doors. Battery charging stands are also of concrete (Fig. 24).

During the period between shifts, and on days when the mine is not operated, fire guards are on patrol throughout the mine, following a designated route. A Detex clock system, with keys placed in all stopes and important smoke-detection points on the levels, is in use. Each fire guard carries a Detex clock and is required to punch his clock at specified times at all keys along his route. The clocks are returned to the Safety Department at the end of the fire-guard day, where the clock charts are checked to see that all stations have been visited at the proper time. At specified times during the shift all fire guards must report by telephone to the toplander at No. 3 shaft, reporting any unusual condition.

Fire Fighting and Mine Rescue

For warning the workmen underground in event of a mine fire, three methods are available: (1) warning stench through the mine by way of the compressed-air lines; (2) warning stench carried into the mine by the ventilation currents; (3) warning direct to the underground force by the use of the mine telephone system.

An ethyl-mercaptan stench apparatus (Fig. 25) has been connected to the main intake compressed-air line at the collar of No. 3 shaft. The box containing the apparatus is in a small room adjoining the collarhouse. This apparatus is charged and always ready for use, instructions being posted in the toplander's office.

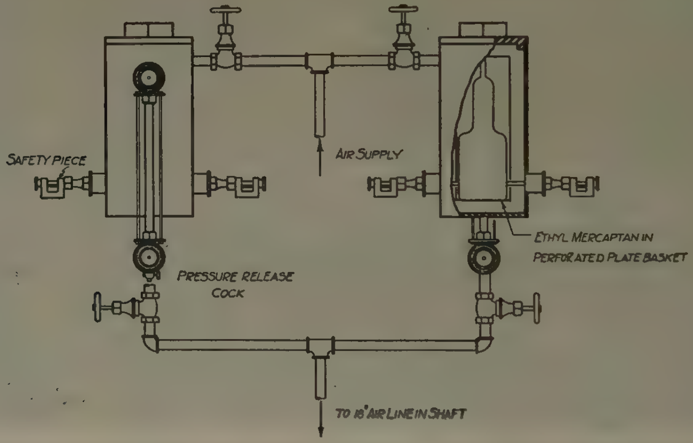


FIG. 25.—ETHYL-MERCAPTAN INSTALLATION AT NO. 3 SHAFT COLLARHOUSE.

Provision has also been made to inject the ethyl-mercaptan stench in the fan intake at the surface of No. 1 shaft. It is planned to use this method only if conditions at the time of the emergency destroy the effectiveness of the stench warning injection in the compressed-air lines.

The mine telephone system serves as an auxiliary method of warning. The system radiates from a switchboard in the toplander's office at No. 3 shaft to the underground supply stations. This mode of fire warning has proved very effective. During fire drills it has reached the men at about the time the stench was first detected. In some cases the telephones proved a quicker method of spreading the general fire alarm.

Refuge stations (Fig. 26) are located centrally in each mining area on all operating levels. They have a minimum size of 8000 cu. ft. and are equipped with air and water lines from the main air and water systems.

A 1-in. perforated air line extends around the outside of the door frame and the valve for its control is in the refuge station. The perforations in the pipe placed around the door frames are pointed so that the air streams converge at a point approximately in the center of the drift.

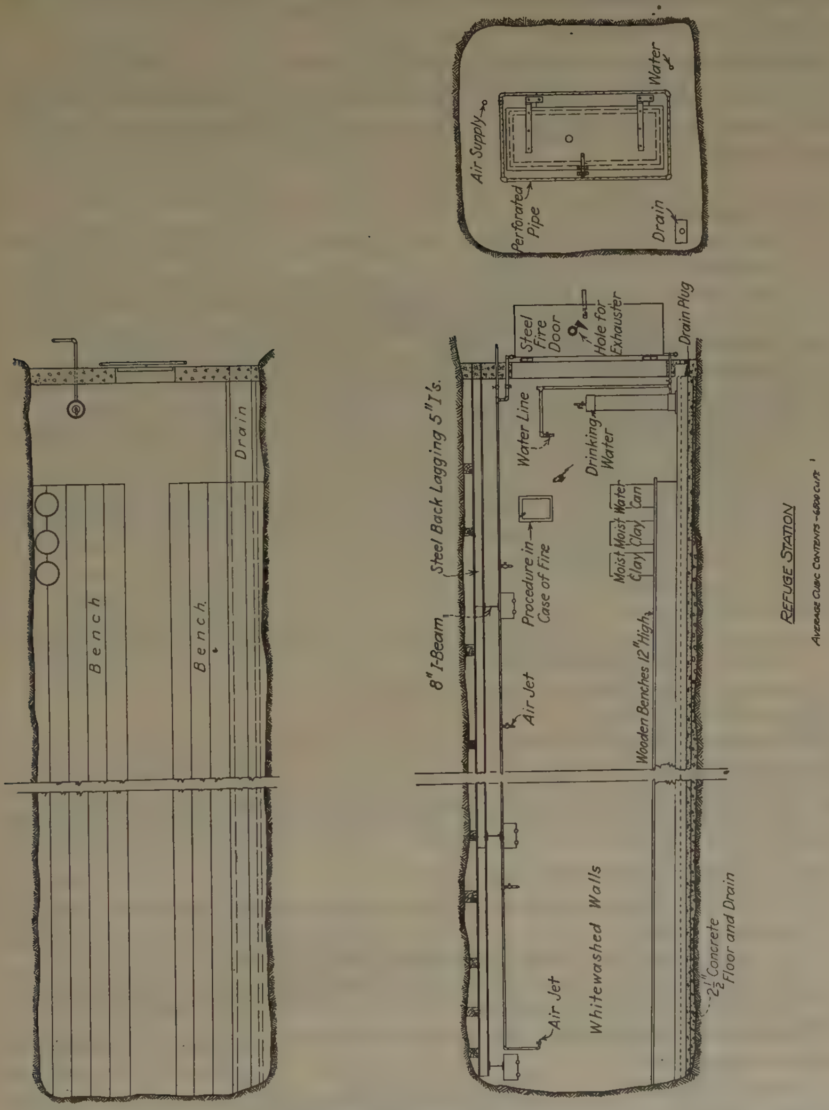


FIG. 26.—PLAN AND SECTION OF STANDARD REFUGE STATION.

Compressed air is turned into this line if gas or smoke approaches the refuge station at the time of the emergency. Two cans of soft clay are kept in the station to seal the door after all the men are inside. In the stations are low benches around the sides, also a standard Bell telephone set. The connecting wires of this telephone system are in the fireproof ventilation shaft. All telephones connect with the toplander's office at No. 3 shaft, for communication between the surface and the men in the refuge station. At ordinary times these stations are used as lunchrooms by the workmen; this policy being adopted to familiarize the men with the location of the refuge stations in each area, thereby preventing confusion in time of emergency.

Regulations governing the general procedure for fire fighting and mine rescue have been adopted, supplemented by a special procedure for a fire in any part of the mine. The general procedure covers the following points:

1. Men discovering the fire:
    a. Shall arrange to notify the No. 3 shaft toplander by mine telephone and shall also notify their level foreman.
    b. If the fire is small and not an electric fire, they shall extinguish it at once, using the fire extinguisher at the nearest supply station.
2. No. 3 Shaft Toplander shall:
    a. Inject the warning stench into the compressed-air line if the fire is not reported to be under control.
    b. Ring nine bells on the mine telephone system and notify all supply stations of the fire and its location.
    c. Notify the first-aid man.
    d. Notify chief electrician or shift electrician to adjust main ventilation fans so that only one fan is operating at half-speed.
    e. Stand by telephone switchboard.
    f. Send cage tenders to No. 1 shaft.
3. Chief Electrician shall:
    a. See to adjustment of fans and arrange for an attendant at fans with instructions to await further orders.
    b. Arrange to have men for emergency work.
4. Master Mechanic shall:
    a. Place hoistman at No. 1 shaft.
    b. Keep compressors running.
    c. Have men available at fans to make adjustment of doors as required.
    d. Supply sufficient surface fire hose for underground connections on auxiliary fire line.
5. Foremen and Bosses underground shall:
    a. Check their men out in the usual way and conduct them into the respective refuge stations.
    b. See that instructions posted in refuge stations are followed.
    c. Conduct their men to the escapement shaft upon receiving official instructions from the mine rescue crew.
6. Fire Doors shall be left open or closed, as directed by the signs on the doors.
7. Mine Rescue Crews:
    a. Shall report to First Aid room for instructions.
    b. Ten men shall proceed to Government Rescue Station, test and equip themselves with oxygen breathing apparatus and other mine-rescue equipment as listed.
    c. Other rescue men shall be governed by orders received at First Aid room.
8. First Aid Men shall:
    a. Notify Government Rescue Station.

  - *b.* Notify Copper Cliff telephone operator, who in turn will notify list of department heads posted in telephone exchange.
  - *c.* Call Mine Rescue men from list posted monthly in First Aid room.
  - *d.* With assistance of watchmen, clerks, etc., notify department heads residing at the mine.
  - *e.* Notify company doctors.
  - *f.* Arrange for necessary First Aid supplies.

9. Headquarters:
  - *a.* Mine Superintendent's office.
  - *b.* Ventilation plans kept here.
  - *c.* Underground plans, compressed-air and water lines kept here.
10. Chief Engineer shall:
  - *a.* Assist in establishing an information bureau at telephone.
  - *b.* Appoint check men to
    - (1) Rope off area at shaft being used as exit from mine.
    - (2) Allow only authorized persons to enter mine.
    - (3) Check in and out of the mine each man by name and number.
    - (4) See that each man entering the mine is properly equipped.
    - (5) Arrange for blueprints of mine plans as required.
11. Supplies Foreman and Stores Foreman:
  - *a.* Shall establish supply headquarters at collarhouse of shaft being used as exit.
  - *b.* Provide necessary supplies as listed.
  - *c.* Arrange for coffee and sandwiches.
  - *d.* Supply housing facilities for Mine Rescue men when off duty.
12. Safety and Ventilation Engineers shall:
  - *a.* Assemble and arrange rescue teams.
  - *b.* Assist in planning rescue and fire-fighting operations.
  - *c.* Arrange for ventilation plans, etc.
  - *d.* Make schedules of approved ventilation changes.
  - *e.* Arrange for emergency hospital, ambulances, doctors, First Aid men, and so forth.

Special fire procedures have been worked out for each shaft and stoping area, dealing mainly with measures for the control and fighting of fire. Instructions regarding movement of the underground force are identical throughout the mine. In all cases the men are instructed to go to the refuge stations. It is felt that this method will insure complete safety. All refuge stations are located in fresh-air currents, and generally are in the main footwall drifts, which are the fresh-air outlets from No. 1 shaft.

For checking men underground, to see that all are out of their respective working places at blasting times, and in emergencies, "check-in" boards and "check-out" boards are placed at the level supply station (Fig. 27). They are about 3 by 4 ft. in size and painted black with horizontal white lines, providing a space for each working place on the level. A row of small hooks is in each space, upon which the men's tags are to be hung. Each man has a brass tag, upon which his employment number is stamped. Before a man enters his working place he must move his tag from the out-board to the in-board, putting it in the space provided for his working place. On leaving his working place for any reason, he must move his tag from the in-board to a corresponding place on the out-board. Space is provided at the right-hand side of the in-board to designate men who remain in the stope to blast.

Each level is divided into areas by fireproof doors, which are fastened in the open position under normal conditions. In case of fire, the doors are closed and tightly sealed with clay.

The fresh-air currents are separated from the exhaust-air currents by fireproof doors, which also serve to control ventilation currents under normal conditions. They will divert smoke and gas from the stope fires to the return-air system. All shafts are enclosed by two sets of fireproof doors, located in the crosscuts to the shafts: one set is placed close to the shaft station in tramming drifts; the other at a distance from the first set greater than the longest motor train. The two sets of doors form an air lock. One set remains closed to air currents at all times.

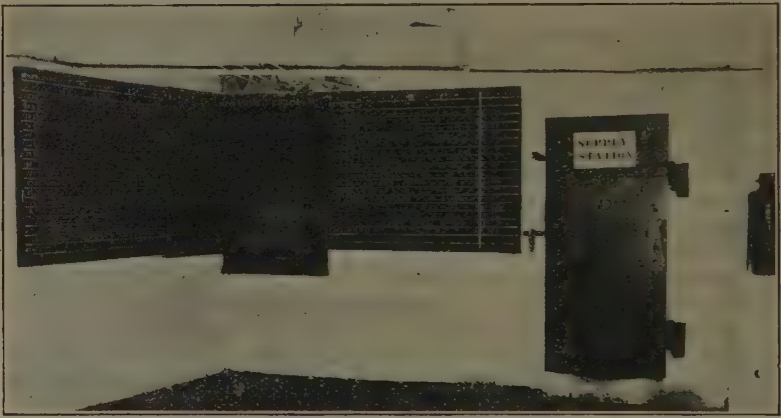


Fig. 27.—Arrangement of "check-in" and "check-out" boards at supply stations.

Fire Hose and Extinguishers.—Fifty-foot lengths of 2-in. fire hose are placed in timbered areas throughout the mine and at all shaft stations. There is a convenient connection for the hose on the high-pressure water line at each hose station. The hose is carried on a reel hinged to a steel frame, enclosed in a moisture-proof box, one end of which is glass and is to be broken in case of emergency.

Pyrene and Fyre Fyter extinguishers are placed at all electrical installations. They are charged with carbon tetrachloride, which is a nonconductor of electricity. Guardene extinguishers are in all underground supply stations. This equipment is inspected and repaired every month and a report sent to the Mine Superintendent.

Water supply for fire fighting is from the high-pressure service for regular mining operations which consists of a 4-in line down No. 3 shaft, and a 2-in. line extending down No. 4 shaft. These two lines are connected on the 2400, 2600 and 2800-ft. levels. Water for levels below the 2800 is taken down the power raise through a 4-in. line, branching off on each level and also connecting with the 2-in. line in No. 4 shaft.

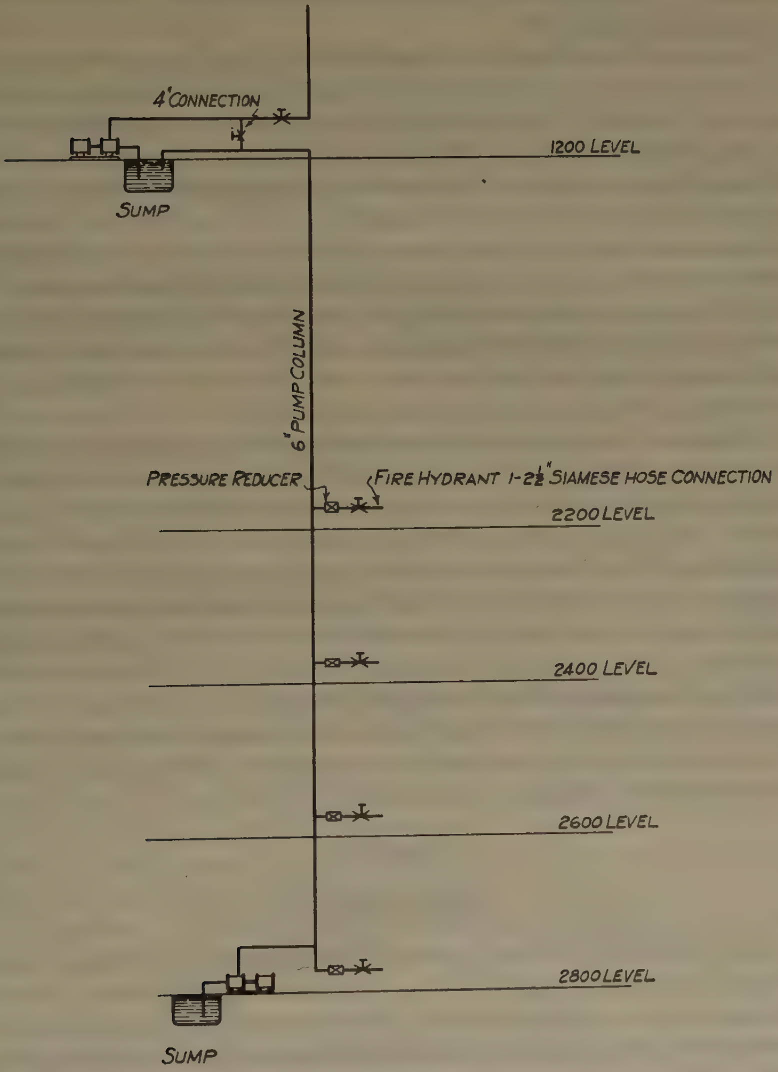


Fig. 28.—Diagrammatic section of auxiliary water-supply installation in No. 3 shaft.

Besides the regular water supply for fire fighting, hose connections have been made to the 6-in. pump column in No. 3 shaft, on the 2200, 2400, 2600 and 2800-ft. levels. Pressure-reducing valves are placed ahead of the hose connections, so that the water entering the fire hose from the pump column is held at 80 lb. pressure. Mine water drains to an 80,000-gal. sump near No. 3 shaft on 2800 level, and is pumped therefrom through a 6-in. column by a 240-gal. per min. centrifugal pump to a sump on 1200 level, whence it is pumped to surface by a similar centrifugal pump. A 4½ by 10-in. single-acting triplex pump is held in reserve at each station (Fig. 28).

In event of a mine fire, the surface fire truck and hose can be taken into the mine, the hose connected to the pump column and run out to the location of the fire. It will be possible, with this arrangement, to return the mine water to the fire. Besides supplying an extra source of water, this installation eliminates the possibility of flooding the pumps during a fire, and insures against a water failure.

Seventeen sets of McCaa oxygen breathing apparatus are maintained at the district Government Rescue Station near the surface plant. In addition to the oxygen breathing apparatus, the rescue station is equipped with 12 All-Service Burrell gas masks, three carbon-monoxide detectors, safety lamps, high-pressure oxygen pump, inhalator, cardoxide, and an extra supply of oxygen for the McCaa apparatus.

A fire truck for underground use, equipped with axes, saws, hammers, nails, brattice cloth, and miscellaneous fire-fighting apparatus, has been placed on a spur track in No. 3 shaft collarhouse and is ready for immediate use. Seven rescue teams of five men each are available for duty in an emergency. They receive regular training in the use of the McCaa apparatus and a problem in fire fighting and rescue is worked out at each training period. All work is done while wearing the oxygen breathing apparatus. In short, an attempt is made in this training to create conditions similar to those anticipated in actual fire fighting and mine rescue. The men in active training on these mine rescue teams are paid at their regular rate for the time spent in training, and receive a bonus of $6 per month as members of the teams.

HOISTING

Mine Signals

The return-bell electric system of signaling is used in all shafts. The bells for each compartment have different tones, so that it is easy to distinguish the compartment in which signals are being given. In hoisting men, there are three distinct signals: first, the three bells indicating men to be moved; second, the signal for the level to which the cage is to be moved; third, the one-bell or two-bell signal for starting the cage. The hoistman repeats each set of signals, to avoid chance of misunderstanding. The code of signals and the regulations governing their use is as follows:

1 Bell —Stop immediately if in motion.

1 Bell —Hoist if at rest.
2 Bells—Lower.
3 Bells—Men about to ascend or descend.
3 Bells—In reply, men may enter cage or other conveyance.
4–1 Bells—Hoist slowly.
4–2 Bells—Lower slowly.
5 Bells—Release. Hoistman may move at his discretion.
6 Bells—Ore.
7 Bells—Rock.
9 Bells—Danger. In case of fire or other danger, follow by signal of level where danger exists.

Signals are to be given only by persons authorized by the mine superintendent. The hoistman, after receiving a signal to move men, shall wait ten seconds before moving the cage. If he has not been able to move the cage within one minute after receiving the signal he shall await a fresh signal. No hoisting shall be done in a shaft or winze where the signaling devices are not in proper working order. During regular hoisting operations, no work on signaling devices shall be begun or carried on that may in any way interfere with their proper operation. A notification shall be given by the electrician to the hoistman when repair work is necessary, and a release when the work is completed.

The three foregoing rules are not intended to tie up a shaft having more than two hoisting compartments, if the signal equipment for one hoist only should go out of order; or if repairs can be made on the signaling equipment for one hoist without interfering with the operation of the other.

Besides the bells, there is a buzzer at each level and at the shaft collar. This does not signal the hoistman, but is used as a call for the cages; it has a distinctive tone and cannot be confused with the bell signals. The emergency signal to be given on a buzzer is:
4–4–4 Accident. Followed by signal of level where accident has occurred. This takes precedence over any other signal.

There is also a telephone system connecting all levels and the shaft collar, from which point telephone messages are relayed by the toplander to the hoistman. The toplander thus becomes responsible for all telephone instructions issued to the hoistman.

Safety Devices

The safety devices installed at the hoist, which operate to insure the safe control of cages are as follows:

Mechanically operated brakes will be applied the moment the power fails. In case of a heavy overload, such as would be caused by a cage leaving the guides or becoming jammed in the shaft, a circuit breaker will cut off the power and thus allow the mechanically operated brakes to come into play. An overwind device, which can be set to engage shaft conveyance at any point in the headframe, will cut off the current in case of an overwind past this point, and thus allow the mechanically operated brakes to come into play. This device must be tested by the hoistman at least once a shift. The brake and clutch gear are so interlocked that it is impossible to release the brake while the corresponding drum is disengaged.

Every hoist has a depth indicator, showing clearly and accurately: (1) the position of the cage; (2) at what position in the shaft a reduction of speed is necessary; (3) the position of a pentice in the shaft. This position to be marked in red.

The indicator is operated by a train of gears from its corresponding drum of the hoist. The indicator dial is a plain surface painted black, the pointer and level markers being painted white. The indicator pointer travels clockwise when the corresponding rope is lowered, and when the cage is lowered to the lowest point in the shaft the pointer does not describe an arc greater than 340°. The pointer travels more than one inch per revolution of the drum.

Inspection Reports

The care and inspection of all hoisting equipment is carried out under the supervision of the master mechanic, who is personally responsible to the mechanical mine engineer for the proper performance of the required inspections. The master mechanic of each mine sends to the mechanical mine engineer daily reports for the hoists and compressors, and a weekly report covering inspection of all ropes, skips, cages and hoists.

Shaft Doors

The shaft doors (Fig. 22) at each level and on the surface are of the swinging type. They are made of heavy sheet metal, reinforced on sides, top and bottom, and diagonally braced. Furthermore, a piece of heavy steel rail is attached to the door about 2 ft. from the level of the track, and running from center to center of the posts on which the door is hung. This reinforcement prevents a moving truck or motor from crashing through the door when closed.

Underground Power, Signals, Lighting Equipment and Telephones

Power for underground service is supplied at 2200 volts, through an oil circuit breaker with relay protection at the collar of No. 3 shaft. The 2200-volt circuit is carried through a 4/0 three-conductor, lead-sheathed, steel-armored, 4400-volt insulated cable to cast-iron junction boxes, located at the No. 3 shaft stations. From the junction boxes current is taken by a similar cable to the operating equipment. For power equipment near the station, the cable is carried in concrete ditches; for inside equipment, it is fastened to the back of drifts by hangers set in holes in the solid rock.

All electrical installations have disconnecting switches and oil circuit breakers, with suitable relay protection. Much of the underground power load is 2200-volt line start motors, some of which are started from oil circuit breakers, others from compensators. The operator stands on heavy rubber mats while starting equipment.

One substation, on the 2600-ft. level, has a reinforced concrete front and a steel door, and is ventilated by an electric fan. The transformers here are the dry type, air-cooled, and have practically no inflammable material in their construction. This type is used underground wherever possible, because of the small fire hazard incurred in their use.

Signals.—The signal system at No. 3 shaft has many new features, incorporated for safety and maintenance of continuous service to the different operating compartments. It is the three-wire common return, taken through the shaft in an 18-conductor, lead-sheathed, steel-armored cable to heated steel junction boxes. These boxes are fitted with push plugs for convenience in disconnecting stations. The power current for the signal system is of 140 volts, transmitted through a double-throw magnetic switch. If the power to one side of the circuit is off, the magnetic switch automatically transfers the feed to the second power source. This insures a continuous supply of power to the signal system. All operating compartments are protected by relays, which, in case of a short circuit in any compartment, will open that section without interruption to the other operating compartments. A voltmeter, ammeter and ground detector are in front of the signal relays. In order to locate trouble on the surface, all operating shafts have a signal system similar to that described above.

Safety Switch Lighting.—In the main tramming areas, where more than one motor train is operated at a time, red and green light signals are used. They are operated by double-throw switches, which are overhead and within easy reach of the motorman. The signals, designed to set a red light against a train entering any drift or crosscut in which another train is operating, are so arranged through a magnetic, double-throw switch that the lights may be operated from either end of a circuit.

Fire and ventilation doors in main haulage drifts are equipped with red and green light signals, so arranged that when the doors are closed red lights warn the motor crew of this fact. The green lights do not come on until the doors are completely opened.

Drift Lighting.—All main drifts and crosscuts are lighted from a three-wire 220 to 110-volt system. Transformers are placed at suitable places in special fireproof stations driven off drifts or shaft stations. In all drift lighting, 100-watt lamps are spaced about 50 ft. apart on alternate sides of the main crosscuts. Reflectors on each light direct its rays to the center of the drift floor.

Transformers.—All underground transformers are housed in fireproof stations, and are enclosed with reinforced concrete front and steel doors. In all oil-transformer stations, catch pits hold the oil in case of a rupture of a transformer. Transformer rooms have ventilation vents in the front wall. A steel plate hinged above the vent hole and held in place by a fusible cord is standard. These links will fuse at a temperature of 160° C., allowing the plate to drop down over the vent hole (Fig. 29).

Transformer primary leads coming into the transformer room are lead-sheathed, steel-armored cable, connected to heavy porcelain primary cut-outs, which are regularly inspected to see that fusing is in good condition and of proper capacity.

Telephones.—Since the three principal shafts are a considerable distance apart, all have telephone systems, operated from a generator-ringing, battery-talking source, with the exception of a magnetic ringing system used exclusively for the underground refuge stations.

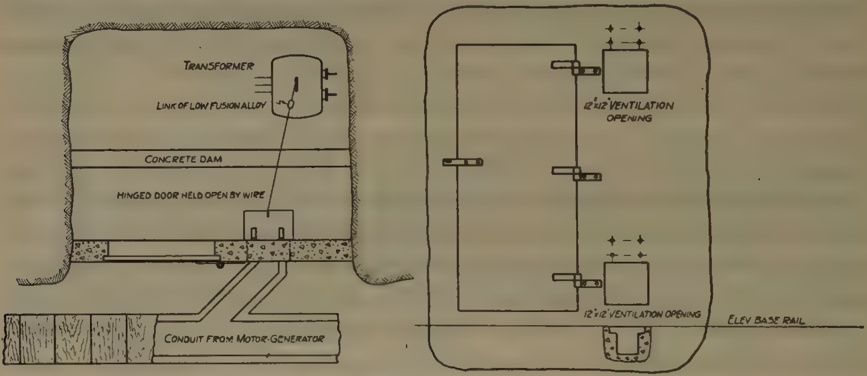

TRANSFORMER
LINK OF LOW FUSION ALLOY
CONCRETE DAM
HINGED DOOR HELD OPEN BY WIRE
CONDUIT FROM MOTOR-GENERATOR
12"x12" VENTILATION OPENING
12"x12" VENTILATION OPENING
ELEV BASE RAIL

FIG. 29.—SKETCH OF FROOD STANDARD TRANSFORMER ROOM.

All systems are connected to a switchboard at the collar of No. 3 shaft, where connections can be made from any section of the mine to any shaft, or through the local surface system to the different shops, offices, or residences of the men living at the mine. The refuge-station telephones may be connected to the Bell telephone system, so that, in an emergency, direct communication can be made with refuge stations from the mine office, general office, or any residence included in the system. This connection is made only on direct orders from the mine superintendent.

Trolley Lines.—The main haulage on all levels is done with 8-ton trolley locomotives. Power to the trolley line is supplied from two 150-kw. motor-generator sets, on the 2400-ft. level.

The circuits leaving the generator station are protected by carbon circuit breakers. Armored cable is used as feeders to the trolley lines.

Each level is protected by an approved circuit breaker, with suitable overload protection. The trolley lines are 2/0 grooved wire, hung on barn hangers spaced 12 ft. apart. The insulators are bolted to steel frames, which in turn are bolted to the timber back or steelwork, and also act as a support for the troughing, which is of 2 by 8-in. fir plank. A clearance of 7 ft. 6 in. between the trolley wire and ball of the rail is maintained. All trolley lines are sectionalized with "Miami" section switches about 1200 ft. apart, so that work may be done on one part of the line without disconnecting the entire level system. At intersections, frogs, etc., pullovers are used, so there will be no displacement of frogs or crossovers when the collector passes over them. All tracks are bonded by gas-welded bonds to the rail flanges.

Electric slushers, fed from the trolley lines, are used in some stopes. They receive power through two-conductor, lead-sheathed, braided steel-armored cable. The ground return is carried through to the slusher. For safety, there is a disconnecting switch at both the slusher and the trolley line.

Accident Prevention

Ever alert to the value of accident prevention, the International Nickel Co. foresaw the need of safety work at Frood while development was yet in the early stages. New mining methods being applied at new horizons, in so far as the camp was concerned, by a newly assembled organization, was undoubtedly a fertile field for accidents. E. A. Collins, appointed General Safety Engineer in 1918, and later promoted to Assistant to the Vice President, carried the burden of the safety work throughout the early development. During the expansion of 1929, Mr. Collins was relieved of this work by the appointment of a new general safety engineer, whose department was formed early in 1930.

The first step was to establish adequate first-aid facilities, with proper recording and reporting of accidents. This done, the safety campaign was launched by forming a central committee, which met once a week for general discussion. This served to coordinate safety efforts, and enable the engineer to lay out his work to the best advantage of all departments. The committee, headed by the superintendent, included the assistant superintendents, general foremen and the safety engineer. Workmen's safety committees were organized immediately to systematically cover the mine on inspection trips. These committees, consisting of three men, one committee appointed from each division, covered their own divisions once every two weeks. The safety engineer acted as secretary to the various groups and copies of their recommendations were sent to shift foremen, divisional foremen and the superintendent. The committee men held office for four weeks, at the end of which new members were appointed. These appointments always went to experienced miners who, aside from their ability to detect unsafe conditions, could advise and instruct new men. However, as standard practices were developed and labor became more stable, the value of this plan dropped so noticeably that it was discontinued early in 1932, and the present system of periodic inspection and report by the safety engineer, coupled with the safety and efficiency bonus system, gradually replaced it. With the ventilation system nearing completion, the department was enlarged to include ventilation, and an assistant safety engineer was appointed to manage this branch of the work.

Safety clothing and equipment has received special attention since the inauguration of the campaign. Hard hats were one of the first items adopted, and by July, 1930, every man working underground was required to have one. The M.S.A. hard hat, with a stiff brim, was adopted as standard.

Goggles have proved to be a vital factor in accident prevention. After considerable preliminary work during 1930, wire-screen goggles became standard for underground use, all being handled by the Safety Department and kept in the first-aid room on surface. If a workman returns a broken or worn-out pair, he is issued a new pair without charge, but if he loses a pair, or his record shows he has obtained an excessive number, he is charged with them at cost. Two main types of glass goggles are in common use. In the surface shops, where protection against splinters from grinding wheels is required, metal-frame goggles with hardened glass of the Willson L. J. 50 type have been adopted. These are also used as a dust and protective goggle by men handling fill on surface. Samplers have been provided with a spectacle type goggle with a side screen.

All underground men must provide themselves with gloves, and the rule is strictly enforced. Frood standards do not specify any one form of glove, but miners are required to wear good leather gloves, purchased through local merchants.

The problem of safety shoes was first attacked in 1930. Types were purchased both in leather and rubber, and numerous tests were made leading to the following conclusions:

1. Miners preferred, as a rule, ordinary high boots, rather than 9, 11, or 14-in. top boots.
2. Leather footwear provided the best protection and was preferable to rubber, except where wet underfoot conditions prevailed.
3. Boots adopted as standard should have safety box toes and heavy counters.
4. There is no advantage in a web tongue.
5. Miners' boots should be heavily hobnailed.
6. For electrical work, there might be some advantage in composition soles.

The management did not feel that stocking shoes in the warehouse could be justified, as this seemed rightly to be legitimate retail business. The Sudbury dealers were therefore asked to submit samples for tests, if they desired to be placed on the list of approved dealers. Samples from four dealers met the required tests and were approved. Although many miners had bought safety shoes previously, it was not until June, 1932, that the order was issued that every man must wear them. Within recent months, for chute pullers, safety spats have been added to the safety-toe shoes. The spat, still more or less in the experimental stage, is of leather, studded with reed slats and heavily padded. It covers the dorsum of the foot, ankle and lower leg, and is of great assistance in preventing metatarsal fractures.

Safety belts are provided and frequently inspected. Each stope is equipped with a belt attached to 30 ft. of ¾-in. hemp rope, which is used when making chute repairs and for general purposes where the 30-ft. limit of extension is applicable. For making long descents into chutes a boatswain's chair is attached to a ⅝-in. marlin-wound cable, the workman wearing a safety belt also attached to the cable. Boatswain's chairs and spools of marlin-wound cable are kept in all underground warehouses, from which they can be drawn for repair work.

Employment

Applicants for work are registered upon forms setting forth their previous experience and other qualifications. The forms are filed in the order of application. The service records of men formerly employed by the International Nickel Co. are investigated, and if their safety records and proficiency at work have not been satisfactory, they are rejected from further consideration for re-employment. Men who have not previously worked for the company are rated according to their intelligence, physical characteristics and experience.

Applicants considered suitable for employment are notified, as required, to call at the employment office. They must first pass a physical examination by a doctor representing the Provincial Compensation Board, including an X-ray examination for defects of the pulmonary system. This is followed by another examination by a doctor of the medical staff of the International Nickel Co., before they are accepted for service. The latter examination is intended to exclude from employment men having defects in vision, hearing, normal movement of arms, legs, hands or feet. In the case of former employees who were injured in the company's service, certain allowances may be made by the examining physician. Careful notation of these are made at the time of re-employment. As a safety measure, it is preferable that all foreigners shall speak and understand English with some degree of proficiency. Having proved to be physically qualified applicants are sent to the mine employment office, where they are entered on the payroll and assigned to a working place.

School of Instruction

A training school has been established for the instruction of new employees and for the further instruction, in any particular type of underground work, of older employees who have proved worthy of advancement. New employees are first assigned as shovelers and fill men. Those who learn the work quickly are advanced to timber helpers and drill helpers, and are assigned, as required, to stopes throughout the mine.

A large square-set-and-fill stope was selected for the training school, which permits the instruction of about 45 men in drilling, timbering, shoveling and filling. The shift boss in charge of the school is selected on the basis of his previous work in the operating department, with especial attention to his safety record. The other instructors are also selected for their outstanding qualities. General supervision of this work falls to the shift boss and his assistant, while the details of the various types of work performed in stoping operations are taught by the stope boss and other instructors. The large staff of instructors permits close supervision of the work, with a correspondingly quick turnover of students, and a maximum degree of safety to inexperienced men.

Student drillers are generally selected from the operating department. They are usually shovelers who have shown particular aptitude and who will probably learn quickly. These men act as drill helpers and blockholers and, as they become familiar with the work, operate the drills themselves and do the blasting. Timbermen are usually drawn from the more competent timber helpers and, therefore, have a good working knowledge of their duties at the start. There is quite a variety of timbering work, and they are instructed in all types before their course is completed. Two slushers are operating in the training stope and new men who have had rigging or mechanical experience are found to be best adapted to learn this work.

In all the work safe practice is especially stressed by the instructors, and it has been found that the number of lost-time accidents to new men has been considerably reduced since the institution of the training school. The men sent to the school by the operating department for instruction as drillers and timbermen, and who do not pick up their new duties quickly, are returned to their former occupations and working places. They are, however, given a thorough tryout before judgment is passed upon them. About one out of six men fails to become proficient in his work, and instruction for an average of 12 working shifts is given before his rejection.

The training school is successfully serving its purpose. Most of the men completing the course have proved competent in their new occupations, and are better grounded in safety practices than those formerly trained in operating stopes. Labor turnover has been reduced by giving further instruction to men who were not satisfactory. Worthy employees have opportunity to advance according to their ability.

Standard Practices

The comparative uniformity of Frood underground conditions have facilitated standardization of work. Underground bosses are supplied with a copy of "Standards," in which are given detail drawings of all timber used in the mine. As new standards are adopted prints of these are distributed. Layouts are made of all new work and prints issued to the foremen soon enough before the work is to be done to enable them to understand the details and plan the work. Instructions for performance of new work are included in the layout.

A copy of standard practices and safety instructions is kept in each underground storehouse, and all bosses are required to sign their names on each page to indicate that they have read and understand its contents. This book contains specific instructions governing the work of bosses, timbermen, drillers, muckers, fillmen, motormen, switchmen, and miscellaneous labor. It is written in outline form in concise and simple language. The bosses in direct charge of men are responsible for the execution of all safety instructions.

To illustrate the form and content of these instructions, some of the more important general items and instructions to stope bosses and leaders are given below.

GENERAL INSTRUCTIONS

1. Special equipment required:
   - *a.* Safety boots must be worn by all men. If the hard toe caps become soft, they may be exchanged for a new pair at the store where they were purchased, provided the rest of the boot is in good condition.
   - *b.* Gloves and hard hats must be worn at all times.
   - *c.* Goggles must be worn by all men while sledging, starting holes, blowing out holes, or any other work where the eyes are exposed to injury.
2. Checking in and out:
   - *a.* Each man is supplied with a small brass tag upon which his employment number is stamped.
   - *b.* When the man is out of his working place, this tag must be hung on the out-board.
   - *c.* When the man goes to his working-place, he must move his tag from the out-board to the in-board.
3. Rules and regulations:
   - *a.* No smoking is allowed in any stope or manway.
   - *b.* All ventilation doors must be kept in the position indicated by the sign on them.

- *c.* Men must not pass between cars when a motor is attached to the train. Always walk around.
- *d.* No "horse play" or scuffling is permitted at any time in the mine.
- *e.* Any injury, no matter how slight, must be reported to the shift boss. When going off shift all injuries must be treated in the First Aid station on the surface. Penalties are inflicted for failure to do this.
- *f.* Toilet cars are provided in convenient places. Ask the shift boss where the one is located in your section of the level.
- *g.* All men must wait at a designated place on the level at quitting time and parade to No. 3 shaft station, where they must line up and enter the cage in an orderly manner.
- *h.* All refuse must be deposited in cans provided for the purpose.

STOPING

Stope Boss and Leader Instructions

1. When going on shift stope bosses must:
   - *a.* Report to shift boss at collar of shaft and receive instructions.
   - *b.* Find out work done by previous shift and plan the work to be done.
   - *c.* Take the first cage to their respective levels.
   - *d.* Proceed to their stopes as quickly as possible. Stope bosses or leaders should be in stope before any work is started.
2. Upon entering stope, each stope boss must:
   - *a.* Inspect manways and have them repaired if necessary.
   - *b.* Inspect guard rails around chutes and dangerous openings, to see that they are in good condition.
   - *c.* Establish a safe travel way if passage under the breast is necessary.
   - *d.* Inspect the mining floor and back lagging, to see that all planks are in place.
   - *e.* Examine the back at newly blasted areas and give instructions to scaler or responsible driller to take down any loose ground.
   - *f.* Examine all blasted faces for missed holes. Blast them immediately if it is necessary to drill within 10 ft. of them.
   - *g.* Instruct shoveler to first wet down and clean the mining floor and then wet down and clean the shoveling floor.
   - *h.* Instruct shovelers or foreman to wet down muck piles.
   - *i.* See that the area in which shovelers are to work is well covered.
   - *j.* Correct any unsafe conditions. If in doubt do not go ahead with work until the arrival of the shift boss or foreman.
3. Laying out work:
   - *a.* Drilling:
     - (1) Show the driller where to set up his machine.
     - (2) Instruct him to scale the breast before starting to drill.
     - (3) See that the open ground between timber and breast is adequately covered.
     - (4) See that the set in which the set-up is to be made is blocked to the back and to the side.
     - (5) Instruct the driller where to drill holes.
   - *b.* Timbering:
     - (1) Show the timbermen where square sets are to be stood.
     - (2) Instruct them to sound the back of the area in which they are to work and to take down any loose ground.
     - (3) Instruct them to erect posts, with headboards on them in the center of each area to be timbered.
     - (4) See that the joints of all square sets erected are tight.

  - (5) See that heavy framed blocks are used on footwall, hanging wall and pillar sets. Lighter material may be used for temporary blocking between sets and stoping face.
  - (6) Examine all wedges to see that they are tight.
- *c.* Shoveling:
  - (1) See that the area in which shoveling is to be done is completely covered, so that shovelers are safe from falling rock or material.
  - (2) See that all muck piles are barred down to prevent rock from rolling on to the shovelers' feet.
  - (3) Instruct shovelers that all large boulders are to be broken. They must not be rolled.
  - (4) See that all powder found in the muck piles is returned to the powder house by a responsible man.
  - (5) See that shovelers do not remove grizzly rails in any case.

General stoping practices:

- *a.* Pillar holes must be drilled outside the pillar row of posts and lined up parallel to the pillar line.
- *b.* Manways and chutes must be constructed according to specifications. Manway clearance at platforms is 24 inches.
- *c.* Gangways on all floors must have guard rails spiked to the sets 3½ ft. from the floor.
- *d.* Ladders between floors must be spiked on both floors and project 3 ft. above the upper or mining floor.
- *e.* Long girts must be reinforced by doubling up underneath them with three false posts; after blasting, remove this auxiliary timber and advance it ahead of the breast.
- *f.* All 3-in. plank must be removed from under the breast before blasting.
- *g.* Until sets have been placed and blocked on the mining floor the shovelers must be protected by placing bulkhead timber on the mining floor. After sets are erected, the bulkhead timber should be used to block the sets to the back and the permanent mining floor placed.
- *h.* The mining floor must not be removed until fill approaches.
- *i.* For stope ventilation the manway compartments in all fill-raise manways must be kept free from obstructions.
- *j.* Holes to be loaded must be cleaned either by blowing out or scraping.
- *k.* Breasts must be scaled before holes are loaded, even though they have been scaled previously.
- *l.* Only authorized men are to handle explosives. When making primers, copper punches must be used.
- *m.* Wooden loading sticks must be used for loading holes.
- *n.* Before blasting, all equipment used in the stope must be removed to a place of safety from the blast.
- *o.* Only 10-ft. fuse may be used for breast holes and must not be trimmed more than 3 ft.; 5-ft. fuses are to be used for blockholes only.

4. Blasting:

Stope bosses must send a selected man down to the shift boss about 15 minutes before blasting time, with a note containing the numbers of men who will remain in the stope to help with the blasting. The stope boss must remain and supervise the lighting of the fuses. The signal to blast will be relayed by the selected man, from the shift boss to the stope boss, after the check board is clear. Not more than 15 holes may be lighted by one man. Two men must be selected to light 15 to 30 holes and 3 men for 30 to 45 holes.

Tramming

For the safe and efficient operation of motor trains on the levels, certain tramming regulations and instructions must be enforced. For this the level boss is directly responsible to the shift boss. Following are the more important instructions to motormen, switchmen, and chute blasters.

1. Special equipment required:
   - *a.* All men who assist in loading ore, rock or fill, must be provided with "spats," which protect the upper part of the foot.
   - *b.* Every switchman must have a whistle for giving signals; also a red tail light to hang on the last car of a train.

2. Instructions to motormen and switchmen:
   - *a.* When going on shift the motorman must call at the electrical shop on surface for the reversing lever of his motor. When going off shift he must return it to the shop.
   - *b.* Before removing the reversing lever the controlling mechanism must be set in neutral position.
   - *c.* In the absence of the motorman, the switchman must check out and return the reversing lever.
   - *d.* At the end of each shift the motorman must make a written report containing the following information in the log book at the electrical shop: (1) condition of the motor at the start of the shift; (2) condition of the motor at the end of the shift; (3) any damage to the motor during the shift; (4) any repairs required to the motor.
   - *e.* If a battery motorman leaves the motor for any reason, he must turn the reverse lever into neutral position and pull the main switch.
   - *f.* If a trolley motorman leaves the motor for any reason he must turn the reverse lever into neutral position and remove the trolley pole from the wire.
   - *g.* Battery motormen are responsible for the charging of their batteries.
   - *h.* Switchmen must not hold couplings in their hands when coupling cars, but must support the coupling in the proper position by using the coupling pin.
   - *i.* In "slow speed" areas the switchman must walk 25 ft. ahead of his train to give warning that the train is coming, and be ready to stop the train in a moment.
   - *j.* In "regular speed" areas the switchman may ride the motor or in an empty car at the end of the train. If riding in an empty car he must crouch down.
   - *k.* Switchmen must see that a red tail light is placed on the last car of each train.
   - *l.* At all stop points the train must be brought to a stop and the switchman must proceed ahead of the train to see if the track is clear before signaling the train to proceed.

3. Code of signals to be used by motormen and switchmen:
   - *a.* One whistle, train shall stop immediately.
   - *b.* Two whistles, train shall go toward switchman.
   - *c.* Three whistles, train shall go away from switchman.
   - *d.* Four whistles, slow signal, and shall indicate the following:
     - (1) When moving at regular speed the motorman shall slow down to 3 miles per hour.
     - (2) When at slow speed the motorman shall go more slowly.

(3) If not moving and four whistles are given, followed by a pause and two whistles, the motorman shall move the train very slowly toward the switchman.
(4) If not moving and four whistles are given, followed by a pause and three whistles, the motorman shall move the train very slowly away from the switchman.

4. General regulations:

a. The maximum speed (regular speed) for motors shall not exceed 6 miles per hour. The "slow speed" shall not exceed 3 miles per hour.
b. All areas are "slow speed" areas for trains transporting explosives.
c. The level boss shall be in charge of trains transporting explosives and carry a red light ahead of the train. The switchman shall walk 25 ft. in the rear of the train.
d. No one shall jump from a moving motor or train.
e. Men shall not pass between the cars of a moving train, nor between the cars of a standing train when the motor is attached.
f. All standing cars and motors must be effectively blocked to prevent a runaway. A piece of ½-in. chain is the standard blocking device.
g. Cars and trucks must be coupled together when being moved by a motor.
h. Motor haulage must be stopped when men are going to or coming from their working places at lunch time and quitting time.
i. Trolley motors must not operate with the trolley pole pointing in the direction of travel.
j. Motors must not go out of their regular working areas except for repairs, battery charge, and to assist an injured man. In all cases the level boss must be notified in advance.
k. When necessary to run on a track where another motor has right of way, the following procedure must be observed: (1) Wait for the motor that has right of way; (2) inform the crew of the motor having right of way that you are about to travel through their area and follow them.
l. No "flying switches" shall be made.
m. All motormen and switchmen must be able to speak English.

Explosives Handling and Storage

The surface powder magazine is at a safe distance from the mine plant. The walls of the magazine are of brick and the roof of reinforced concrete in the form of a low arch. The magazine is heated by circulating hot water, of which the temperature is controlled by an electric thermostat. The heating and control units are housed in a separate building. A continuous temperature record is maintained by a recording thermometer. The temperature records show a minimum temperature of 40° F. in winter and a maximum of 65° F. in summer.

Powder is received at the mine in carload lots, stored in the magazine and issued on mine requisitions. On receipt of the daily explosive requisitions the powdermen remove the powder from the original packing. The oiled paper and sawdust are taken out of each case and the powder replaced in a box. A copper-edged wedge and wooden mallet are used to open the powder boxes, and the lids are replaced after the paper and sawdust have been taken out. The amount required for each day is opened and cleaned on the morning of the day the powder is sent into the mine. The powder to be delivered to the underground magazines is loaded in a standard explosives car, which is cylindrical in shape and is made of sheet metal, lined with wood. There are two sliding doors on one side. All moving or rubbing parts are separated by brass rivets welded into the sheet metal. Each car has a capacity of 33 cases. These cars are designed to run on standard-gage mine track (Fig. 30).

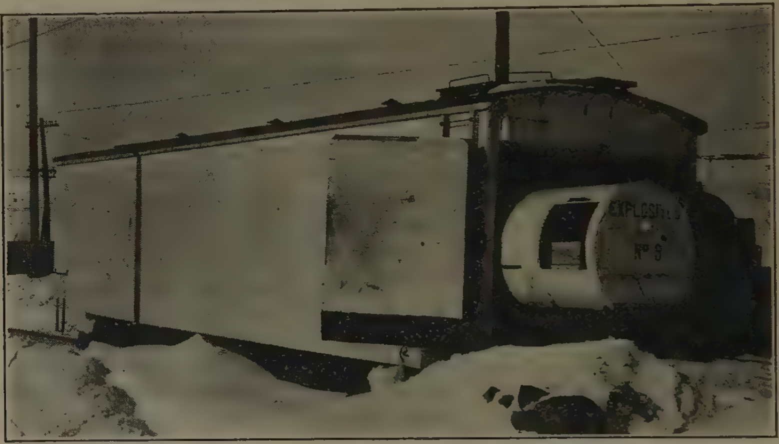


FIG. 30.—POWDER CARS USED FOR TRANSPORTATION OF EXPLOSIVES.

The loaded underground powder cars are pushed by hand out to the platform adjoining the standard-gage spur track, where they are loaded into a standard-gage box car fitted with double track. The tracks in the box car are of rubber belting, secured to the floor with copper nails. Rubber car-stops prevent the powder cars from shifting in the box car. One end of the box car is fitted with doors that open out, and after the box car has been loaded with the explosives cars the doors are locked. Duplicate keys for these locks are held by the powdermen and the surface fusehouse men.

Each morning the powder car is shunted by an electric standard-gage locomotive to an unloading platform at the mine. After the powder has been transported to the underground magazines the explosives cars are returned to the surface, where they are replaced in the standard-gage powder car and the doors are relocked. The car is returned to the surface magazine for the next day's loading.

All fuses are cut and capped in a surface fusehouse, located 710 ft. from the collar of No. 3 shaft and 235 ft. from the nearest building. The fuse reels are placed on spindles at one end of the fusehouse. Three fuses at a time are drawn from the reels and cut into 10-ft. or 5-ft. lengths. All fuses are capped with No. 8 caps, using a California type fuse cutter and cap crimper. After the fuse has been cut and capped, the ends are dipped in a red or green dye solution, according to the length of fuse. The tips of the 10-ft. lengths are dyed red and the tips of the 5-ft. fuses are dyed green, to eliminate possibility of a 5-ft. fuse being used in a round of holes with 10-ft. fuses, and to avoid the possibility of a fuse being trimmed twice. After drying, the fuses are tied in bundles of 50. The 10-ft. fuses are used in development headings and stope breasts, while the 5-ft. fuses are used for blockholes.

Fuses are packed in special fuse cans for transporting underground. These are made from carbide cans, lined with cedar, including their hinged covers. Two bundles of 10-ft. fuse and one bundle of 5-ft. fuse are packed in each can. The cans are placed on a timber truck for transport into the mine.

Explosives are lowered into the mine each working day on a schedule outlined by the supply foreman. When explosives are being loaded on the cage, an electric switch is closed in the toplander's office, which flashes red lights at the hoist and at all shaft stations. The red lights warn the men that powder is being lowered and to keep clear of the shaft until the operation has been completed. Regular lowering and hoisting signals are used.

The explosives are guarded at the underground shaft stations by the station tenders until taken away by the level boss. A level motor crew accompanied by the level boss transports the explosives to the level powder magazines. This is done at lunch time on the day shift, when regular level haulage is stopped. The underground powdermen receive the powder at the underground magazines, and check the amount received. The fuse is stored in a separate magazine.

Underground powder and fuse magazines are usually located on main footwall drifts, and must be at least 200 ft. from any underground shaft, winze or raise. They are about 8 by 7 ft. in section and driven 10 ft. into the side of the drift. The fronts of the magazines are of concrete and have metal-faced doors. A small hole is left on one side of the doorframe for ventilation and inspection. They are locked at all times, and only the powderman is allowed to enter them for issuing powder.

Some of the regulations for handling, transport and storage of explosives in the mine are as follows:

1. Powder in transport to underground magazines must be in charge of some responsible person at all times.
2. All underground powder and fuse magazines shall be in charge of a responsible person designated by the mine foreman and supply foreman.
3. Explosives shall not be stored underground in excess of the necessary supply for 24 hr. In no case shall an amount exceeding 300 lb., or six cases, be stored in any one magazine.
4. Fuse, blasting caps and electric detonators shall not be stored in a magazine with powder, but in a separate magazine not closer than 25 ft. to a powder magazine.
5. Open lights are not permitted in the magazines.

Powder and fuse are requisitioned from the level magazines by the level bosses and stope bosses for each working place. The requisition is filled by the powderman. Powder and fuse are carried to the working places in separate bags. Any powder or fuse not used is returned to the magazines and credited to the working place. The requisitions are kept by the powderman until the end of the shift, when they are turned in to the Mine Accounting Office.

Safety Inspections and Reports

The Safety Department consists of: (1) Safety and Ventilation Engineer; (2) Assistant Safety and Ventilation Engineer; (3) First Aid men.

The Safety and Ventilation Engineer is responsible to the Mine Superintendent and to the General Safety Engineer. He enforces "Standard Practices" with the aid of the supervisory force on the surface and underground; he is in charge of underground ventilation, operation of the fans, and aids the engineering department to plan new work in this connection. The Assistant Safety Engineer spends most of his time on ventilation. He makes a complete temperature survey twice a month and a volume survey four times a year. Local volume changes are made as their necessity is indicated by the bi-monthly temperature survey.

The Safety Engineer inspects all working places in the mine at least three times monthly, sending a written report to the general foreman, foreman and shift boss of any irregularities in practice. An oral report to the shift boss or foreman is made at the end of the inspection, and any criticism open to controversy is reviewed on the scene when possible. These reports are called "General Appearance" letters, and are criticisms of the working places as well as of adherence to the standard practices. Each working place is rated on these inspections; a perfect one is rated at 25 points. Points are deducted from 25 for each infraction of safety regulations, or standard practices. This rate is used to compute Safety Bonus at the end of each month. The rate allotted to the shift boss is an average of those allotted each working place. The foreman's rating is an average of his shift bosses' rating.

While the stope bosses, leaders and level bosses do not participate in any bonus except the regular production bonus, a rating is computed each month for each sub-boss on "general appearance." The purpose of this is to create a spirit of competition between them, and to furnish a check on their observance of standard practice. Their fitness for their job depends largely on their accident record and "general appearance" rating, as well as their production efficiency.

Inspections are made of all places where serious accidents occur and reports are made to the superintendent and general foreman. Any accident that might result in serious injury is investigated. These may or may not be reported, depending on circumstances.

Sprays for dissipation of dust at ore dumps are inspected by a member of the mine pipe crew at least twice a week. Samples of drinking water in the mine are taken weekly and sent to the Ontario Department of Health for analysis.

A daily accident report is issued to the foreman of each underground division, containing the following information: (1) working place in which the man was injured; (2) name of stope boss or level boss; (3) nature of accident; (4) total number of dressings done at the First Aid room, (5) remarks on discipline.

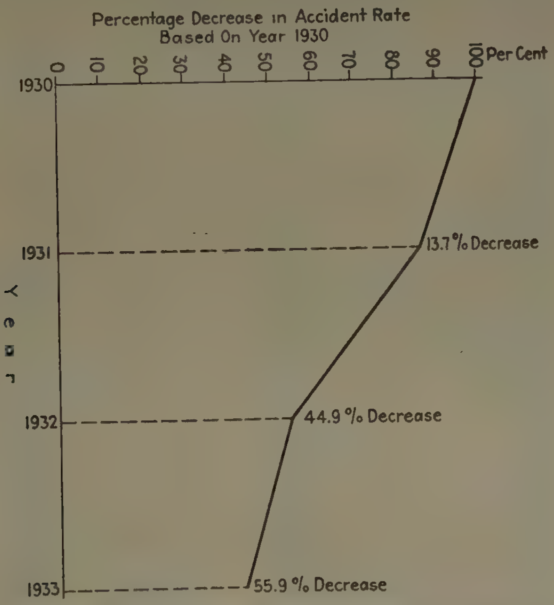


FIG. 31.—PERCENTAGE DECREASE IN ACCIDENT RATE BASED ON YEAR 1930.

In the monthly report, accidents are classified as to cause and severity. Each shift boss, foreman and general foreman is rated for the month, the year to date, and for minor lost-time and compensation accidents. These reports are ammonia prints, and are issued to the superintendent, assistant superintendents and all foremen. The final report at the end of the year gives the ratings for the year of all men included in the report. It consists of a classification of all lost-time accidents as to the occupation of the injured, type of injury and seat of the injury; it is made on an ammonia print. An occupational hazard and severity rate is computed from this report (Figs. 31 and 32).

During the year, reports of all serious accidents and underground fires, required by the superintendent or the general safety engineer, are made by the safety and ventilation engineer. Through the general safety engineer, he also reports when required to the Ontario Inspector of Mines and Workmen's Compensation Board.

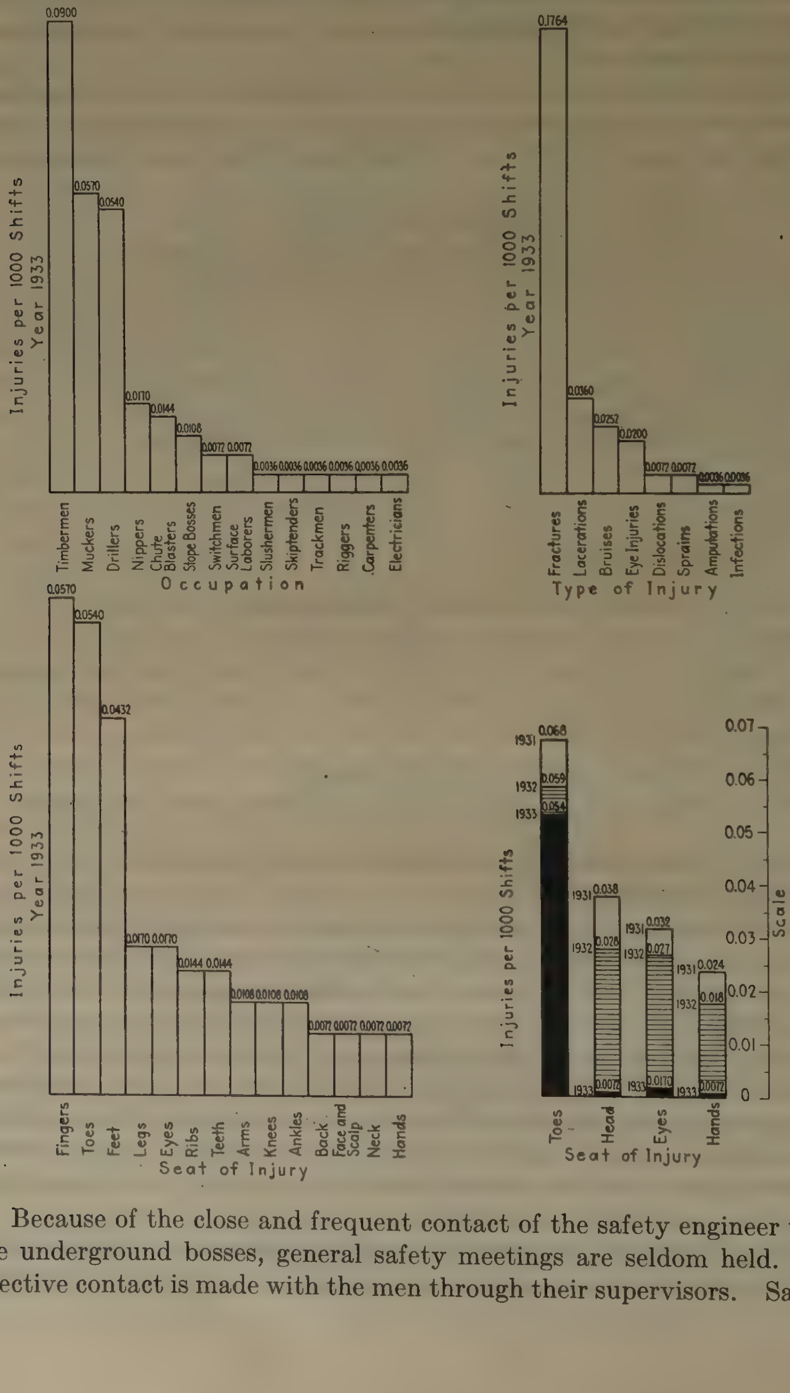


FIG. 32.—ACCIDENT FREQUENCY FOR YEAR 1933.

Because of the close and frequent contact of the safety engineer with the underground bosses, general safety meetings are seldom held. An effective contact is made with the men through their supervisors. Safety is taken up with the foremen and superintendent, in conjunction with other operating problems, at meetings held once a month.

In the collarhouse at No. 3 shaft a large board, upon which are painted clocks with movable hands, shows the safety record of each division. The dial numbers represent shifts worked. The large hand points to a number showing the total shifts worked in each division since the last lost-time accident, the small hand indicating the best previous record (Fig. 33). Placed next to this board is another, on which the names of all shift bosses are listed. Opposite each name is a green and a red electric light bulb. The green light is kept burning until a lost-time accident occurs; it is then replaced by the red light. This furnishes a visual monthly record of lost-time accidents.

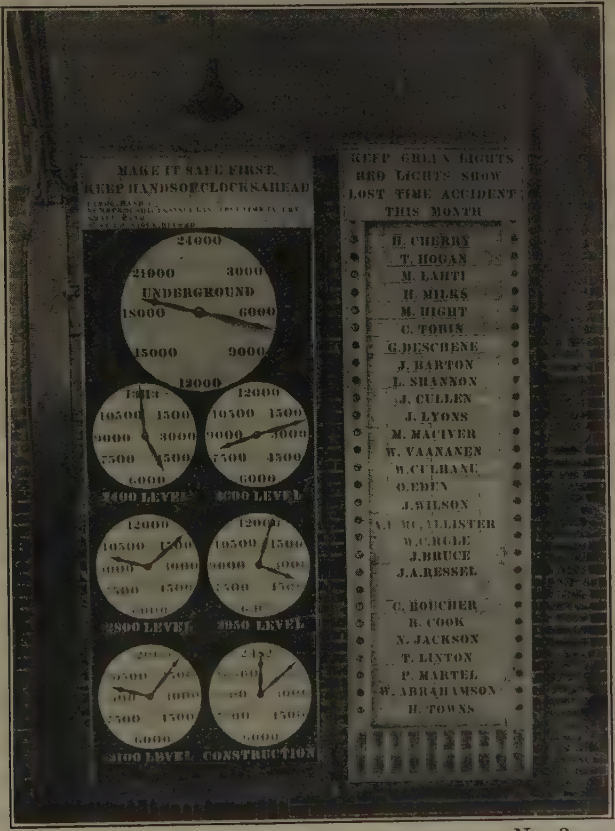


Fig. 33.—Safety-record boards at collarhouse, No. 3 shaft.

To carry out effectually a policy of standardization requires a certain amount of disciplinary action. Deviation from safe practices on the part of the men is punishable by lay-off or discharge. The general practice is to warn men after the first offense, lay them off from two days to two weeks for the second offense, and discharge them for the third. Records of disciplinary action are kept on the men's accident cards. Men discharged as poor safety risks are interviewed by the safety engineer, who may modify the punishment if it appears that discharge is too severe. Men returning to work under these circumstances are placed on probation and given only a short period in which to make good. After the safety engineer has passed on a case, the man may appeal to the superintendent if he feels he has been unjustly treated. The superintendent's decision is final.

Bonuses

A bonus system has been developed, by which underground foremen and shift bosses are rewarded for efforts in promoting safe practices and efficient work. This system has given excellent results. Its competitive features tend to sustain the interest of the foremen and bosses in accident prevention, creating keen rivalry between them in the matter of safety results.

The bonus is paid each month to the foremen and bosses with the highest ranking. The ranking is based on accident prevention, general appearance and efficiency. The total possible rating is 100 points; 50 points being allowed for accident prevention, 25 for general appearance and 25 for efficiency. The amount of bonus paid each month for the highest rating is as follows: senior foreman, $35; junior foreman, $30; shift boss, $30; shift boss with second highest rating, $20; shift boss with the third highest rating, $10.

Rating for Accident Prevention.—The possible points are 50, from which 10 points are deducted for each lost-time injury occurring in the area under the supervision of the foreman or shift boss concerned. Obviously some bosses have more men under their supervision than others, so that to make the competition fair it is necessary to compute the rating for each boss on the same number of labor shifts. This calculation is based on the greatest number of labor shifts under any one boss for the month. For all other bosses enough shifts are added from the previous month to make up the total, and if any lost-time injury has occurred in this period it thus affects their rating for the month.

For example, in making up the shift boss rating for a certain month it is found that shift boss A has the greatest number of shifts, say 2000. Shift boss B has had only 1500 shifts under his supervision, so it is necessary to check back into the preceding month to make up the remaining 500 shifts. If in this part of the preceding month one of his men has sustained a lost-time injury, 10 points are deducted. Assume that there have been two lost-time injuries among A's men during a month. His accident rating is 50 less 20, or 30 points. If B has had no injuries during the month, but had one in the preceding month, his rating would be 50 less 10, or 40 points.

Rating for General Appearance of Working Places.—The possible points are 25. At frequent intervals throughout the month the safety engineer inspects each area. Any infractions of safety regulations and unsafe conditions and practices are noted and reported by letter to the superintendent, foremen and shift bosses, which shows the rating granted and reasons for deductions. Thus unsafe conditions and practices are brought to the attention of the supervisors and bad procedure is quickly remedied. The criticisms are made in a constructive way and the foremen give careful attention to their correction.

On about the twentieth of each month a rating report for the period is posted, showing the bosses in order of ranking for general appearance of working places. At the end of the month the rating of each boss is considered by the superintendent and assistant superintendent in conference with the safety engineer.

Rating for Efficiency.—The possible points are 25. Rating is measured by the amount of work done per labor shift, compared to the approved standards set for all underground work. The standards are based on a large number of careful studies made by the Contract Department. Each competitive area has a standard efficiency allotted and bosses receive perfect rating when they obtain this efficiency. Standards are expressed in tons per labor shift for stoping, development, tramming and miscellaneous labor.

Posting.—At the end of the month a rating sheet is posted. It shows the points awarded under each classification and the order of ranking of the bosses for total points. Copies are sent to each boss and are posted in prominent places about the plant.

The condition or cleanliness of working places is considered fundamental in any successful attempt at accident prevention. Hence it is felt that the bonus as outlined here has a most important effect on safety. The inclusion of the efficiency feature in a safety bonus, while a novel idea, helps to keep a balance between safety work and efficiency.

First Aid and Medical Attention

First aid to the injured is given due consideration. A fully equipped station, with trained men on duty 24 hr. of the day, is maintained on the surface. This station is a center for all first-aid activities and is directly under the supervision of the safety engineer.

The three first-aid men are holders of St. John Ambulance Association certificates and work in close relation to company doctors. They are instructed to administer only such help to injured men as comes under the scope of first aid.

Shift bosses, stope bosses and underground warehousemen are given periodical training in first aid, which is conducted by the regular staff of first-aid men and is under direct supervision of a company doctor. A certificate of merit, in the form of a card, is given to all men who pass the prescribed examination at the end of the course. To sponsor this work the company provides that all payroll men taking the course shall receive a bonus on successfully completing the examination.

All underground storehousemen are first-aid men, and it is their duty to administer first aid to injured men before sending them to the station on the surface. A well equipped first-aid kit is kept at all warehouses. At all shaft stations and underground warehouses there are boxes containing a stretcher carrier, woollen blanker, 50-ft. length of ¾-in. rope, splints and bandages for splints. They are painted white and an electric light within them is kept burning.

Three company doctors are stationed at Sudbury and to them falls most of the work in connection with the Frood mine. The doctors pay periodical visits to the mine to inspect the first-aid equipment, and one is always on call in case of accident.

Hospital facilities are provided for accident cases in a well equipped, fireproof company hospital at Copper Cliff. This building will accommodate 40 patients, but by furnishing extra wards the number could be increased to 60. All modern hospital devices and conveniences are provided, including operating room with monel-metal tables, X-ray department, sun rooms and a dispensary for out-patients. The staff consists of the Chief Surgeon, four assistant surgeons, and eight nurses. The company maintains an excellent motor ambulance, which is on call 24 hr. a day.

For reporting and recording accident cases, forms are used. When an accident occurs the shift foreman is notified by the injured man or by fellow workers. If the accident is serious the injured man is sent to the surface first-aid room. The first-aid clerk notes in a record book information covering all details of the accident, including the man's employment number, name, place and time of accident, time of reporting, description of the accident and names and numbers of any witnesses. He then makes out a notice of accident for the doctor. This form is on 4½ by 7-in. paper bound in duplicate in books, with the original perforated. Information on the form gives the man's name and number, the time and place of accident, and the first-aid clerk's diagnosis. The original of this is given to the man to take to the doctor. From the record book the permanent record form is compiled and forwarded to the general safety engineer at Copper Cliff. This is a large form on 8½ by 14-in. paper, with the lower 4 in. of the form perforated. Besides the particulars of the accident, this form contains all necessary information as to length of service and earnings, to enable proper returns to be made to the

Workmen's Compensation Board of Ontario. The lower perforated portion of the form is for information regarding the return of the man to work. This is detached and held by the first-aid clerk until the man returns, when it is completed and forwarded to Copper Cliff.

Company doctors have bound duplicate-form books 9 by 7 in., the original of which is perforated to the binding and also horizontally through the middle. On first examination of an accident case the doctor fills out the upper part of this form, known as the "Doctor's first report," which gives the man's name and number, time of reporting, description of the injury and prognosis of disability. This is forwarded to the general safety engineer at Copper Cliff. When the doctor finds the man ready to return to work he fills out the lower part of the report, which serves as official notice to the safety engineer that the man is able to return to either light or regular work, as specified by the doctor.

Correspondence with the Workmen's Compensation Board is carried on through the office of the general safety engineer at Copper Cliff, where the permanent files are kept. Permanent accident files are indexed on a duplicate card system, according to accident number and alphabetically according to name, for statistical purposes. The index cards are 4 by 5 in. and contain the following information: (1) name and employment number, (2) claim number with W.C.B., (3) file number, (4) nationality, (5) marital state, (6) occupation, (7) department, (8) foreman, (9) previous claims, (10) prognosis, (11) date injured, (12) date returned, (13) shifts lost.

After careful investigation of an accident the Frood safety engineer sends a form notice to the shift foreman, setting out certain particulars according to information received from the workman, and asking the following questions:

1. In your opinion what was the cause of the accident?
2. How could it have been prevented?
3. Where do you place the responsibility?
4. What steps have been taken to prevent accidents of this type in future?

The first-aid office keeps a card record of all employees, on which information is noted regarding each accident suffered by an employee. This serves as a check on accident breeders and facilitates the weeding out of this type of workman. The cards are 5 by 8 in., lined, and are headed with the man's number, name and occupation. Vertical columns provide for the following data: (1) date of accident, (2) hour, (3) accident number, (4) shift foreman, (5) reports, (6) lost time, (7) place, (8) injury, (9) cause, (10) returned.

Rehabilitation.—In most cases, injured men returning to work are not able to assume their original occupations. They are grouped under the heading "light duty," and put in charge of the safety engineer who

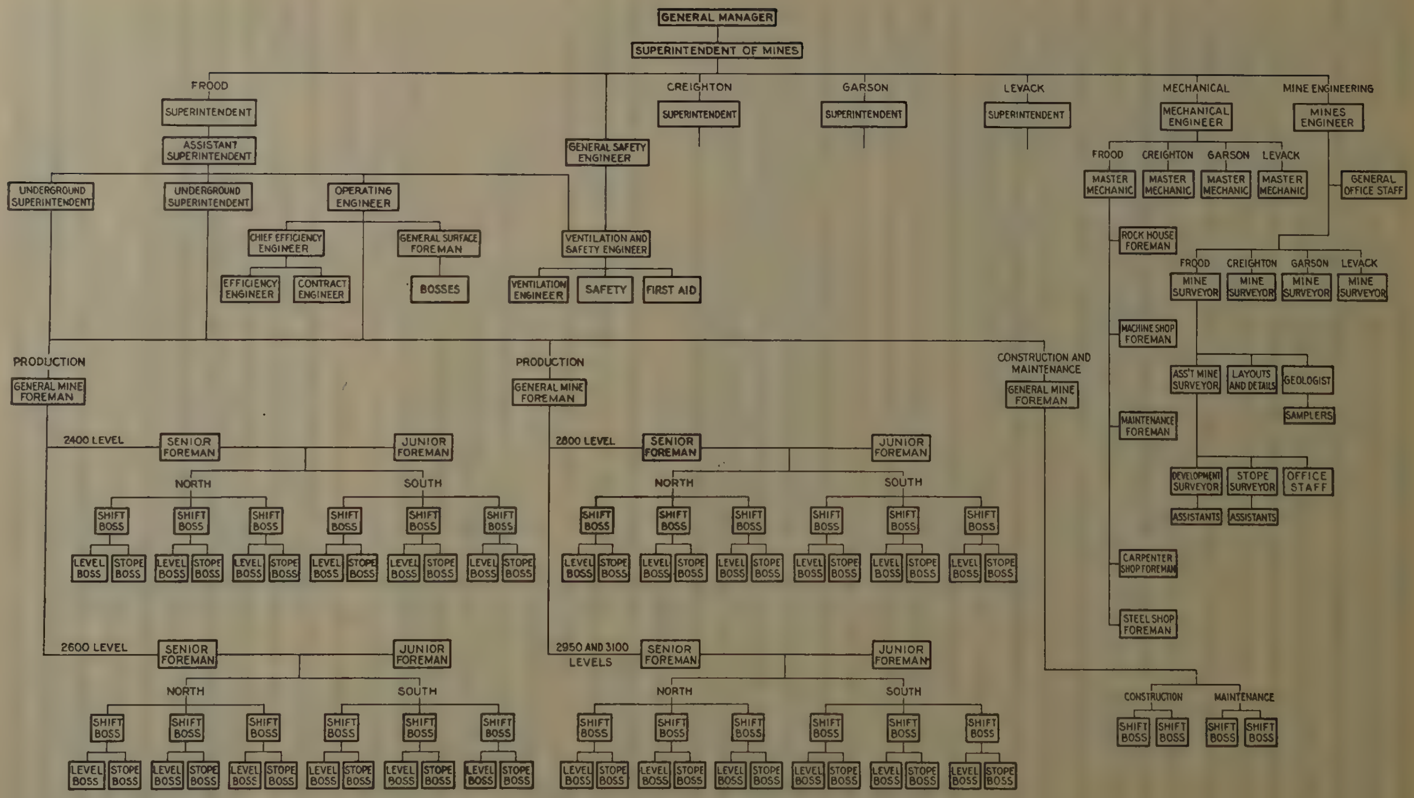


FIG. 34.—FROOD MINE PERSONNEL.

places them on such duty as cleaning, painting, whitewashing and ventilation maintenance. The job selected for an injured man depends on the nature of the injury, with a view of allowing him to perform some needed service while convalescing. Usually, the occupation must be such as will cause the injured man to exercise joints and muscles to restore their normal action.

Men who have become too old in the service for hard physical labor are given lighter work until they are pensioned. A number of jobs that can be done satisfactorily by these partial disability cases are kept open.

PERSONNEL

The personnel of the Frood mine engineering, mechanical, surface, and underground staff is given in Fig. 34.

ACKNOWLEDGMENTS

The preparation of this article was made possible through the collaboration and generous cooperation of the Mining Department of the International Nickel Co. To all of these men, I am most grateful.

DISCUSSION

(*J. T. Ryan presiding*)

A. S. Richardson,* Butte, Mont.—What is the reason for parallel fan operation?

R. D. Parker.—Duplicate fans were installed in parallel primarily as insurance that one fan would always be available for operation and continuous ventilation maintained in the event of repairs, inspection, etc. It also permitted a wide variation in fan capacity.

A. S. Richardson.—Were gases found in the mine that required continuous fan operation?

R. D. Parker.—The only gases originating and so far experienced were products of blasting, and continuous ventilation was maintained in order to keep temperatures down.

A. S. Richardson.—What was the rate of change in air temperatures at different seasons of the year, and were fan capacities varied?

R. D. Parker.—Table 3, which I could not read in presenting the paper, on increase of temperatures at depth for varying velocities, answers that question. Fan capacities were of necessity varied in extreme conditions in order that air delivered to the levels should not be below a freezing temperature.

C. M. Smith,† Urbana, Ill.—Why were not the hoisting shafts used as intake or return airways?

R. D. Parker.—The entire ventilation system was planned so as to keep all main travel ways in fresh air. Upcasting of the timbered shafts would hasten the decay of timber and expose employees to dust-laden air as well as increase the resistance against the fans. Downcasting of main hoisting shafts in a country that experiences long periods of extreme cold is impracticable because of the formation of ice on the guide runners and shaft timber.

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* Ventilation Engineer, Anaconda Copper Mining Co.

† Research Associate, Department of Mining Engineering, University of Illinois.

D. Harrington,* Washington, D. C.—There are many excellent features in this mine, which is one of the best ventilated mines in the world. Not many coal mines approach it in this regard. Probably no other company has as well planned and executed its operations, both as to ventilation and ore production. In spite of having a large mine employing many men working in ore high in sulfides, with much timbering, the fire and other hazards are kept at a minimum. In case of fire, men on any level can reach fresh air by passing through only one or two doors. Although dust is not an imminent hazard, the company is in advance of others in dust prevention.

Why are gloves worn by the men; what protection is obtained from their use?

R. D. Parker.—The men are required to wear unlined leather gloves of either the gauntlet or plain type. Their main protection is from minor cuts, bruises and infection from ores and materials.

A. S. Richardson.—What temperatures and humidities give the highest working efficiency?

R. D. Parker.—From observations to date the objective of the Frood ventilation system as regards air conditions is to limit the temperature around the working places to from 65° to 72° F. with a relative humidity of about 80 per cent and a reasonable air flow of about 125 ft. per min. The standard aimed at is equivalent to a wet kata cooling rate of 19.0.

G. S. Rice,† Washington, D. C.—What type of lights are used, and is smoking permitted underground? Why do timbermen suffer a disproportionate share of the accidents?

R. D. Parker.—We are now using electric cap lamps and electric tail lights for transportation. We use an improved fuse lighter and prohibit smoking in stopes, main gangways and crosscuts. The electric lamps are working satisfactorily and it will be increasingly necessary for metal miners to overcome their prejudice against that type of light. As to accidents to timbermen, they suffer the most exposure to the friable ore, as they go ahead of everyone else and make the workings safe for those who follow.

J. T. Ryan,‡ Pittsburgh, Pa.—The Frood mine is the largest nickel mine in the world and may be the largest copper mine.

F. A. Jordan.—Is Frood a larger producer than Creighton? What percentage of quartz is found in the ore?

R. D. Parker.—Until 1930 Creighton was the larger but it is now serving primarily as a rock producer to supply fill to the Frood mine. There is a negligible amount of quartz in the Frood ore.

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* Chief Engineer, Safety Division, U. S. Bureau of Mines.

† Chief Mining Engineer, U. S. Bureau of Mines.

‡ Vice President and General Manager, Mine Safety Appliances Co.

Economic Aspects of Silicosis

By Benjamin F. Tillson,* Upper Montclair, N. J.

(New York Meeting, February, 1934)

Abstract†

THERE is a dearth of information on the economic threat of silicosis. Even the insurance companies and rating bureaus are in a quandary, and the majority of them avoid any action that will stir up an awakened public consciousness of the serious nature of this industrial hazard. Individual claims for $100,000 are now common in silicosis lawsuits. Because of "hidden costs," in lowered plant efficiency, the total cost of this disease to industry is probably four times its apparent direct cost. The direct costs vary with the labor policies of the employer and with the labor legislation in force. Direct costs may originate in three ways: (1) voluntary assumption by the employer of medical expenses and loss of earning power of stricken employees; (2) state enforcement of industrial disease compensation acts; and (3) awards or settlements after suits in common law. Examples are offered in the last two classifications. In the Rand, South Africa, compensation laws are in effect and in approximately 20 years prior to 1932 about $65,000,000 has been awarded for silicosis and an additional liability of $34,000,000 incurred, or an average of about $5,000,000 per year. During 1932 silicosis compensation on the Rand was about $6,000,000. No figures are available for the direct capital or operating costs for preventive measures there, nor for the additional "hidden costs." It might be reasonable to assume that silicosis has cost the gold-mining industry of South Africa twenty to twenty-five millions of dollars per year. It is estimated that common-law silicosis claims have, in the last few years, aggregated $50,000,000 in New York State alone and have exceeded $300,000,000 in the United States.

Attention is called to the involved nature of the insurance situation for industrial diseases in contrast to accidents. The indeterminant, insidious nature of silicosis makes it particularly difficult to predict necessary insurance premiums and reserves. The serious hazard of this disease was not properly appreciated by either insurance companies or industrialists, so sufficient study has not been given its causation, prevention, detection, treatment, and compensation. Therefore legal difficul-

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* Consulting Engineer.

† The paper was published in *Engineering & Mining Journal*, June, 1934.

ties have arisen in respect to the coverage intended by the standard insurance policies that have been in force.

The magnitude of the economic threats of silicosis and the suddenness with which it has descended upon employers and insurance carriers are evidently beyond their conception or preparation, even though the terms of the policies and the premium loadings were intended to cover occupational diseases. Perhaps it is so serious that, to avoid bankruptcy and dislocation of industry, social legislation must be restrained from giving complete protection to individual workers in order best to serve the interest of society as a whole.

Pathological Reaction to Various Mineral Dusts

By Leroy U. Gardner,* Saranac Lake, N. Y.

(New York Meeting, February, 1934)

Abstract†

The human respiratory tract is an apparatus for the interchange of gases between the air and the blood. An important part of it are mechanisms to prevent the apparatus from becoming clogged with dust particles that are inhaled with the air. These mechanisms are adequate to protect the organ against most dusts, but silica and at least one of the silicates, asbestos, are tissue poisons which initiate changes in the connective tissues of the lung, incompatible with function.

Silica is concentrated by the phagocytes in contact with the connective tissues of the lymphatic drainage system. An overgrowth of connective tissue occurs which interferes with the function of this system. More dust that may be inhaled can no longer be effectively eliminated from the lung. It accumulates in the functioning part of the organ and excites the formation of characteristic fibrous nodules and bands. These changes seriously impair the respiratory function. The shadows of such changes cast upon an X-ray film of the lung are discussed. The reaction to silica in the lung produces a marked increased susceptibility to tuberculous infection. It can be reproduced and studied in the experimental animal and has been observed repeatedly in human beings.

Asbestos dust also causes fibrosis in the lungs but the mechanism of its development is different. The fibrous dust settles along the irregular walls of the terminal bronchioles. In this location it excites the formation of cuffs of scar tissue. This contracts and shuts down the tubes, resulting in a collapse of the air spaces distal to the obstruction. Such a condition in itself is a cause of fibrosis. The two causes, immediate reaction to the dust and collapse of the finer air spaces, bring about a generalized diffuse fibrosis. This reaction is not so regularly associated with increased susceptibility to tuberculosis as with silicosis.

The case against sericite as a cause of silicosis has not been proved. Experimentally, sericite-free quartz will produce nodular fibrosis. Silicon carbide and aluminum oxide are quite inert in the normal animals. The former apparently increases susceptibility to tuberculosis in animals but whether the same is true in human beings awaits demonstration.

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† The paper was published in *Safety Engineering*, March, 1934.

Meeting the Dust Problem When Drilling Rock

By R. A. Brackett,* Hartford, Conn.

(New York Meeting, February, 1934)

Abstract†

The most effective means of controlling the dust in the drilling of rock has been found after careful investigation to be collection of this dust at the source. By direct collection, all the dust is collected at the collar of the drill hole and deposited into suitable containers. Equipment for handling this work can be either portable or stationary, depending upon the particular requirements and the application. For most work the portable unit is found most advantageous, and with a capacity capable of handling two drills operating simultaneously.

Such a portable unit would consist of a metal cap, or hood, placed against the face of the rock through which the drill steel would be inserted. This hood is connected by a flexible rubber hose to a primary separator, where the bulk of the dust is collected. From here a second length of hose connects to the secondary stage of separation, where the fine particles of dust are collected and which is built in as an integral part of the vacuum producer.

Hoods are adapted for vertical, horizontal or stope drilling, and mounted so as to become immediately effective when collaring the hole and remain in place continuously.

Such dust collectors can be used with either wet or dry drilling. However, in view of the fact that when drilling dry and using the dust collector, the dust count can be maintained within safe limits, wet drilling becomes unnecessary. Furthermore, it has been found that when using the dust collector not only has the dust hazard been eliminated, but also increased footage can be obtained.

Samples taken under a wide variation of conditions indicate that where the dust count required is below five million particles per cubic foot, dust collectors will be required. Operators, in general, accept the use of this additional equipment favorably, and it is generally felt that such means of collecting the dust will soon be universally adopted.

Tests made by the Connecticut Department of Health Bureau of Occupational Diseases show dust counts reduced from 203 million particles to 6.05 million by the use of the dust collector, when drilling dry.

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* Sales Manager, The Spencer Turbine Co.

† The paper was published in *Safety Engineering*, March, 1934.

The Diagnosis of Silicosis

By Adelaide Ross Smith,* New York, N. Y.

(New York Meeting, February, 1934)

Abstract†

Both history and physical examination are unreliable in the diagnosis of silicosis. In some studies 25 per cent of individuals with silicosis have been found to have no symptoms whatever.

The positive physical findings that may be produced by this disease include diminished chest expansion, diminished lung resonance, distant and harsh breath sounds. However, many workmen of a short stocky type have diminished chest expansion without any lung disease. In a study of rockdrillers made by the author more than half of the men without silicosis were limited in chest expansion to 2 in. or less instead of the "normal" 6. In a study of stonecutters the average chest expansion in the negative group was found to be only 2.4 inches.

Changes found upon physical examination of the lungs themselves are so inconstant as to be of little diagnostic value. In the studies of rockdrillers and stonecutters already referred to, approximately 25 per cent of those with advanced silicosis apparently had normal lungs.

The usual functional tests of lung performance are likewise of little value because of marked individual variation. Average vital capacity will be reduced in a group of silicotics below what might be expected in a group of normals, but certain individuals of the former group may have an entirely vital capacity, giving no indication of the underlying presence of silicosis.

It follows that the X-ray examination is absolutely indispensable in the diagnosis of this disease. X-ray findings characteristic of silicosis have been described innumerable times. They consist essentially of a mottling of the lung fields due to the presence of fibrous nodules which develop as a reaction to the presence of the irritant silica.

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* New York State Department of Labor, Division of Industrial Hygiene.

† The paper was published in *Safety Engineering*, March, 1934.

Industrial Dust Sampling and Analysis

By Leonard Greenburg,* New Haven, Conn.

(New York Meeting, February, 1924)

Abstract†

The American literature in the field of dust sampling and analysis has been growing rapidly since 1915. Studies made since that time clearly indicate that there are three fundamental factors that determine the production of lung pathology due to dust inhalation. These are the duration of exposure, the chemical and mineralogic nature of the dust and the concentration of the potentially harmful dust.

In the estimation of the amount of the potentially hazardous dust there are essentially three problems involved: (1) a determination of the size of the hazardous dust; (2) determination of the most satisfactory apparatus and technique of sampling, and (3) the most satisfactory method of analysis.

A complete study of the literature shows that industrial dust varies in size from ultramicroscopic particles to those larger than 10μ in diameter and that relatively few particles are found over 5μ in diameter. Approximately 10 to 25 per cent are less than 0.8μ in diameter and the median size as a rule is found to lie between 1.25 and 1.75 microns. From the point of view of lung pathology the particle sizes of most importance lie between 0.5 and 5μ.

Of all the instruments for sampling dust the Impinger apparatus is the only one that will sample a larger body of air satisfactorily with small control errors and high efficiency in both low and high concentrations of dust. The Owens apparatus has the advantage of being more portable and somewhat easier to use but it does not permit of the sampling of large volumes of air. It has a low efficiency in high concentrations of dust and does not permit dust weights to be obtained.

From the point of dust-counting technique the author points out that in 1922 considerable experimentation with dark field and oblique illumination resulted in the conclusion that such methods were not readily duplicated from day to day and therefore yielded somewhat erratic results. Changes in the amount of illumination produced variations of several hundred per cent in dust counts obtained with dark ground

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† The paper was published in *Safety Engineering*, March, 1934.

illumination and made it quite apparent that transmitted light was the best to use for routine dust-counting work.

The technique previously outlined serves to obtain all dust particles between 0.7 and 0.8μ and 10μ as an upper limit. Probably small amounts of material, 10 to 20 per cent, below 0.8μ may be lost but the advantages of a simple reproducible technique more than compensate for this difference in results, since no method yet devised yields any more than relative results.

Economic Notes on the Nonmetallic Mineral Industries*

By Paul M. Tyler† and Oliver Bowles,‡ Washington, D. C.

(New York Meeting, February, 1933)

The extensive employment of nonmetallic minerals antedates the use of metals, but only within the last two decades has the production of nonmetals begun actually to keep pace with the complicated activities which characterize the so-called Machine Age. The economic changes ushered in by the Industrial Revolution were reflected first in an increased demand for metals and for fuels.

In the modern sense, therefore, the nonmetallic mineral industries are new and afford new opportunities for engineer and economist alike. A certain amount of caution is indicated, therefore, before attempting to apply ready-made formulas to the problems of these industries. Economics is not an exact science. In the field of production technology where the number of variables is fairly well known, the principles of economics are readily applicable, but as the student proceeds through the province of marketing into the unexplored regions of psychology and sociology he must distinguish carefully between scientific reasoning and purely metaphysical speculation, even when the latter masquerades as economics. The present paper attempts to prove no preconceived theorems and is designed primarily to stimulate careful thinking on sundry neglected phases of these industries. An attempt is made to differentiate the general problems into single aspects which afford promise of being comprehended separately; also, some fundamental statistical data are presented, but merely to indicate a method of scientific approach applicable to more intensive studies.

Definition

Before discussing the economics of the nonmetallic mineral industries, it is appropriate to consider what we mean when we say "nonmetallic minerals." This term, inasmuch as it is merely a negative description, is somewhat loose and ambiguous and gives rise to questions in regard to minerals that, like chromite or bauxite, are alternatively classed either as nonmetallic or as metallic minerals (or ores). The chrome ore used to make ferrochrome for steel manufacture is undoubtedly metallic in character; similar material destined for use as a refractory in the same industry is by common consent considered as nonmetallic. Even limestone is a potential source of calcium, a metal which recently has given promise of finding moderately important applications. Feldspar, it will be remembered, is a complex silicate of aluminum and other elements, including potassium, sodium, and often lithium—elements which the chemist considers as extraordinarily metallic in character. It is not generally known, for example, that the tonnage of sodium metal annually produced (from common salt) is substantially greater than that of mercury, magnesium, molybdenum, bismuth, cadmium, cobalt, gold, silver, or sundry other commercial metals.

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* Published by permission of the Director, U. S. Bureau of Mines.

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Methods of production now provide a less satisfactory means of distinction even than chemical composition. Formerly the ores were won primarily by underground mining, whereas the nonmetals were taken from open cuts; however, power-shovel operations in the copper and iron-ore industries and numerous examples of underground quarrying of building stone, cement rock, and clay have combined to make such a generalization obsolete. Moreover, owing to a freer exchange of engineering skill, there is no longer any sharp demarcation between the equipment used for such operations as crushing, screening, classifying, and even concentrating; machinery firms now cater to the trade of both metallic and nonmetallic mineral producers, offering the same equipment in many instances.

In the final analysis the definition resolves itself essentially into a criterion of use. Minerals that are not used primarily as sources of metals are considered as nonmetallic and those used mainly as fuels—notably coal and petroleum—are considered separately from other nonmetallic minerals.

General Classifications

The nonmetallic minerals other than fuels may themselves be subdivided into groups on the basis of use, although only in a general way. Building construction constitutes by far the dominant outlet. Agriculture consumes the products of the fertilizer industries which are broadly grouped with other chemical industries employing a variety of nonmetallic mineral materials. The smelting, refining, and remelting of metals cannot be accomplished economically without the employment of numerous "nonmetallic" fluxes and refractories, which for convenience may be classed as a metallurgical group.

Pottery, glass, and the closely allied enamelware industries are almost exclusively dependent upon nonmetallic mineral raw materials, and this is true to an even greater extent of the important industries that manufacture brick, tile, and other heavy clay products. The consumption of natural abrasives such as garnet or corundum has tended to decline, but the tonnage of bauxite and sand used in the production of artificial abrasives has grown rapidly. In the paint trade, pigments of nonmetallic origin compete with and supplement metallic derivatives, and a somewhat similar condition exists in the rubber industry. In sundry special ways nonmetallic minerals are employed in the electrical industry (mica), papermaking (clay and talc), automobiles (asbestos brake linings), oil refining (fuller's earth), lubricants (graphite), and in a great many other branches of industrial activity. The piezoelectrical properties of quartz—to take a highly technical example—are utilized in the radio industry, and the optical properties of several minerals are utilized in the technical control of numerous industrial processes such as sugar manufacture.

While tentative groupings are possible, attempts to classify these minerals, even on the basis of use, may result in confusion, inasmuch as many of them are used in a number of different industries. Limestone, for example, is employed as building stone; is crushed for concrete aggregate or for fluxing stone; is pulverized for agricultural use, for dusting coal mines, or for a filler in many diversified products; is mixed with clay and burned into cement; or is burned alone into lime which in turn is variously employed. In the present paper, therefore, little attempt has been made to distinguish subclasses except to segregate the major group of building materials from the other nonmetals.

SIZE AND IMPORTANCE OF THE NONMETALLIC MINERAL INDUSTRIES

According to Bureau of Mines statistics, approximately one-half of the total value of mineral production of the United States represents mineral fuels (coal, oil, and gas). The remainder is about equally divided between metals and nonmetals, although the tendency has been for the nonmetals to gain faster than the metals group. In 1930, the value of mineral fuels produced in this country was estimated at $2,764,500,000, the value of metal production at $986,975,000, and the value of the nonmetals at $1,013,325,000. In value of production, therefore the nonmetals exceeded the metals by a considerable margin in 1930. Under the abnormal conditions of 1931, the total values of all groups were greatly reduced, but the nonmetals attained a higher relative level, exceeding the metals by nearly 30 per cent.

In terms of volume, the production of nonmetallic mineral products usually is well in excess of 400 million tons a year, considerably more than double the ore output of metal mines, and more than 10 times the annual tonnage of pig iron and nonferrous metals.

Mere bigness, of course, is not the sole measure of importance from a national standpoint, but it can be readily shown that this diversified group of mineral materials can easily hold its own from the standpoint of utility and general industrial and national significance. Even in the highly stressed field of military importance, the essential character of such items as sulfur, mica, or cement is universally recognized. Without limestone, lime, gypsum, phosphate rock, nitrates, or potash, impoverishment of the soil would soon result in national disaster. Chemists call lime "the king of all the bases," and trade in sulfuric acid is rightfully watched as an industrial barometer, so universal are its applications. Were it not for stone, sand and gravel, cement, lime, gypsum, and ceramic products, our construction industries would be paralyzed and our transportation systems would break down for lack of highways, railroads, bridges, and dams.

Economic Services

An intelligent study of any industry necessitates an analysis of the nature of the services rendered by its products. Are the products durable or, like fresh fruits and flowers, do they soon vanish from the market? Are they necessities, comforts or luxuries? Is the demand a natural urge or is it an acquired taste or temporary fashion? The distinction between producers' goods and consumers' goods is often significant and such obvious matters as general availability of similar or substitute products, transportation costs, and a host of special considerations all need to be pondered before the businessman or the student can proceed rationally to plan ahead or even comprehend existing conditions. For so diverse a group of commodities as the nonmetals, however, the discussion is necessarily couched in general terms.

Broadly speaking, the nonmetallic minerals are raw materials and must be joined with other materials and labor added before they reach the ultimate consumer. As is well known, industries that produce goods of this character are generally subject to greater fluctuation in demand and speculative activity than industries that deal more directly with the ultimate consumer.

The bulk of these commodities being classed as materials for building construction, the services which they render are essentially durable in character. The life of an office building is usually at least 20 years, a residence may last a lifetime, and even the barite in a coat of paint is there to stay for several years. The householder has to buy food and fuel, and when the ice box and the coal bin are empty, he generally manages to replenish them fairly promptly. But when it comes to building a new house or making extensive repairs to the old one, he usually waits until circumstances or a favorable attitude of mind impels activity. This psychology, exaggerated often by the leverage of purely speculative building, tends to lift the booms even higher and to depress the declines even lower. Durable materials are therefore most subject to booms and depressions. They may, however, be sustained to some extent by such artificial means as extensive programs of public works and highway construction.

Salvage of used materials is another phase of durability. More efficient methods for collecting and utilizing scrap have brought hardships to producers of virgin metals—hardships that are especially burdensome when depression causes a shrinkage in total demand. Producers of nonmetals are more fortunate in this respect. There is a market for used brick, old stone structures are torn down and the stone reused, and old paving stones are recut, but these examples are merely exceptions to the general rule. Moreover, the adverse market influences of these reclaimed materials is softened by the fact that the tearing down of old buildings generally occurs at times when new building is active and the market can readily absorb the old along with a much larger amount of new material.

Diversification in Use

Diversification in use is a stabilizing influence because the aggregate demand for a product that has many uses in different industries is normally well sustained, whereas the demand for products that depend for their outlet solely on customers in the same kind of business may fluctuate greatly. Lime is of the diversified type, for it is used in agriculture, in a multitude of chemical processes, and in the building trades. Theoretically losses in business in one consuming industry should be offset by gains in another, and consequently lime production should normally be well sustained. The fact that activity in this industry failed to keep pace with that in many other industries during the prosperous years prior to 1930 was due to the extraordinary competition lime encountered both in agriculture and in the building trades. That diversification does tend to maintain stability is borne out to some extent by the sand and gravel, crushed stone and cement industries, which, during the recent business stagnation, have made up in highway construction much of the ground lost in building.

Competition

In many lines of business the old rivalries between manufacturers of the same product have been largely supplanted by alliances to fight the invasion of markets by substitute materials. In respect to its effects on the nonmetallic mineral industries as a whole, competition from materials of other than nonmetallic mineral origin is of importance mainly in the fields of roofing and flooring materials. Interproduct competition is of vital importance, but generally it concerns two or more products within the nonmetallic mineral industries group. With respect to a particular contract, conflict is more likely to arise between limestone and sandstone, for example, than between stone and wood. All-metal construction has not introduced real competition, at least as yet. Or the other hand, compromises such as those involving the use of thin veneer of stone or brick with a backing of concrete (or other cheaper nonmetallic reinforcement) are common and may be described as competitive turning points in the development of certain industries. Also, certain communities of interest exist within the industries. Cement cannot be used without aggregates—whether sand and gravel, crushed stone, or slag—and the use of aggregates would be greatly limited without cement. Sand and lime, or mortar and brick, represent similar linkages. Most important of all is the fact that almost every modern building calls for a variety of nonmetallic minerals, glass, plaster, and concrete entering even into frame structures.

TRANSPORTATION

The most costly of all commodities, with the single exception of radium, are the gem stones, but most of the nonmetallic mineral substances are fairly abundant in almost every locality and are therefore relatively cheap. An intermediate group (including such items as asbestos, mica, china clay, graphite, and fluorspar) is less widely distributed, of higher unit value, and hence enjoys a moderately wide or even a world market; but workable deposits of the large-tonnage items (stone, sand and gravel, and clay) are ubiquitously common and consequently possess mainly what in the language of the economists is known as "place value." A deposit of poor clay near a market, for example, may be worth much more than a deposit of good clay further away. Common brick, unless it has an exceptionally pleasing color, must be sold close to its point of origin; otherwise the freight rate to market will prevent successful competition with more favorably situated establishments. A ledge of trap rock that would be highly valuable in a populous and growing American community would be worthless if situated in the savage wastes of Africa. In brief, the value of many deposits is created solely by near-by markets, and as the radius of distribution advances outward from a given plant it soon reaches the point where the delivered cost of the product, including transportation charges, far exceeds the cost of opening and operating a new deposit which ordinarily can be found close by.

PRINCIPLE OF NATURAL SELECTION

More important even than transportation considerations is the fact that the physical properties of many nonmetallic mineral materials cannot be changed by smelting or other beneficiation or purification processes similar to those by which the metals are made marketable. They have to be used practically as nature made them, and all the producer can do to improve the quality is to pick and choose. Notwithstanding recognized methods for bleaching barite, improving the whiteness and texture of paper clays, and the various devices of the ceramic chemist, the salability of a given product is dominated largely by the whim of nature and can be modified only within narrow limits, if at all. It is clearly impracticable to melt up a granite block and recast it; if it contains a defect the buyer must either accept it or demand a new block. The waste piles around most dimension-stone quarries bear mute witness to the practical significance of this simple principle.

It follows that too rigid specifications and insistence on qualities over which man has no control except by selection may work a hardship on both producer and consumer. The latter by acquiring an attitude of sympathetic understanding of the producer's limitations may frequently save himself much of the needless expense engendered by rejecting at the quarry (or even at point of delivery) a large percentage of material which he might use merely by making some minor modification in his own plant processes (or prejudices).

Supply and Demand

With respect to most commodities, the statement credited to Ricardo is correct: Men err in their production; there is no deficiency in demand. Commodities are produced primarily to sell. When they cannot be sold, production stops unless someone takes the responsibility of financing the accumulation of stocks. Hand-to-mouth buying has always characterized the trade in nonmetallic minerals, and, owing to the bulky nature of most of these minerals, abnormal accumulations in producers' yards loom large enough to discourage continuance of production for more than a brief interval after consumer demand drops off. In the sand-and-gravel industry, for example, errors in forecasting demand are more likely to be expressed in excessive plant capacity than in a surplus of production over consumption. The sulfur companies normally carry huge stocks, but these are made necessary by the methods of production and hence in principle do not represent an exception to the general rule that production of the nonmetals tends always to keep closely in step with consumption. However, potential supplies of most of the bulky products, mainly building materials, are so abundant and so readily made available that they tend to exert upon the market many of the characteristic effects of large stocks.

At the risk of seeming to insist upon the obvious, it is proper to mention the inexorable operation of the law of supply and demand as discussed in any standard textbook on economics. One classical example is the air we breathe; few things are more needful to our existence, but it costs us nothing because the supply is superabundant. On the other hand, mere scarcity does not create commercial demand unless the goods themselves satisfy a want. Few people would pay a large price for masurium, alabamine, or virginium, notwithstanding the fact that these elements are in such scant supply in our world that they were not even discovered until quite recently. The manifold applications of these essential principles to the nonmetallic mineral industries individually or as a group should be kept constantly in mind.

BUSINESS CYCLES

Inasmuch as production of most of the nonmetallic minerals is sensitive to changes in consumptive demand, production trends in one industry are rarely countercurrent to those in other industries. The magnitude of the changes may differ, but the flow tends to be always in a common direction. Certain groups of commodities, moreover, tend to move even more closely together than commodities in general. Various indexes are available as thermometers of business activity and (either directly or indirectly) production, both for industry as a whole and for certain groups of allied industries. Index numbers, or percentages calculated upon some arbitrary base (usually actual figures for a recent year or group of years) are the usual measures, but Ingalls[1] has adopted pounds per capita as a means of showing production trends.

It is from such data that we learn that the weight of a major depression does fall most heavily upon producers' goods, including materials for construction (*i. e.*, durable goods); to a lesser extent on fuels; and least of all on foods and clothing. It is noteworthy that in our present "machine age" the bulk of the population of the United States is able to buy—even at a reduced price—very nearly its normal needs of food, fuel, and clothing (*i. e.*, consumers' goods), whereas most economists are agreed that in the recent depression the prices of raw materials have declined more drastically than in any previous depression of record, and this is partly true likewise with respect to the production of these materials.

CONSUMPTION TRENDS IN THE UNITED STATES

For specific nonmetallic mineral industries annual domestic data for available supplies (production plus imports minus exports) have been calculated into indexes which may be compared with production indexes for other commodities. These indexes are presented in two groups; Table 1 covers building materials and Table 2, miscellaneous items. It is generally known that most industries suffered further declines in 1932 (for which year data are not yet available) and it is not known when (or if) the long-delayed upturn occurred. Inasmuch as the pattern of the present trough is not yet clearly outlined, the present paper discusses the tabulations in only a preliminary way.

Building Materials.—The demand for building materials as measured by the value of total building contracts awarded (F. W. Dodge figures

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[1] W. R. Ingalls: Changes in Consumption of the Chief Commodities in the United States. *The Annalist* (July 22, 1932) **40**, 107.

TABLE 1.—*Building Materials Consumption Indexes*, 1910–1931
(Average Volume of Materials 1923–1925 = 100)

Year	Industrial Production[a]	Building Contracts Awarded[b] (Value)	Concrete[c] (Pavement)	Dimension Stone	Cement	Aggregates[d]	Heavy Clay Products			Slate	Gypsum	Lime
							Common Brick	Face Brick	Hollow Building Tile			
1910	66	e	e	e	53	e	126	33	e	67	46	83
1911	58	e	e	e	54	e	116	34	e	62	45	80
1912	70	e	e	e	58	e	117	38	e	92	48	83
1913	73	e	e	e	61	e	100	39	e	63	51	85
1914	60	e	e	e	59	54	97	38	e	57	48	80
1915	73	e	e	e	59	57	94	40	e	54	48	85
1916	91	e	e	100	65	57	101	47	e	44	54	96
1917	90	e	e	80	62	49	80	35	e	42	53	89
1918	87	e	e	45	48	37	49	17	e	25	40	76
1919	80	63	e	59	58	45	65	37	e	57	47	78
1920	85	63	e	62	66	53	66	37	67	65	60	84
1921	64	56	e	62	65	57	62	41	52	57	55	60
1922	84	79	86	77	80	67	80	66	75	85	74	86
1923	101	84	86	95	93	88	99	90	98	98	91	96
1924	95	94	100	97	100	101	98	95	92	101	98	96
1925	104	122	114	108	108	112	103	115	110	100	111	108
1926	108	129	117	99	111	117	103	113	107	101	114	107
1927	105	129	137	103	118	130	96	112	107	97	109	104
1928	108	135	162	101	120	134	87	112	103	90	108	105
1929	115	117	152	104	116	140	75	100	95	93	106	101
1930	93	92	159	97	109	133	50	69	73	62	77	80
1931	78	63	147	68	87	105	32	43	51	51	58	64

[a] From data of Standards Statistics Co. (Inc.).

[b] Federal Reserve Board unadjusted index (from F. W. Dodge Corporation).

[c] Indexes calculated from monthly averages, thousands of square yards, Highway Bureau, Portland Cement Association.

[d] Total tonnage building and paving sand and gravel, and crushed stone used in concrete and road metal.

[e] Data not available.

calculated as three-month moving averages by the Federal Reserve Board) remained at a relatively low level during the World War, and while building was fairly active during the depression of 1920–1921, it was not until 1924 that the index rose above the production index for commodities in general. An all-time peak was reached in 1928, but the downturn in building contracts anticipated that in general industrial production by almost a year and subsequently the slump proceeded further. Public building continued for three years to be close to the record levels reached in 1928 but failed to offset the appalling shrinkage in contracts let for (predominantly) private construction, which dropped from over 4½ billion dollars in 1928 to about 1½ billions

TABLE 2.—*Domestic Consumption Indexes for Specified Minerals Used in Chemical and Manufacturing Industries*, 1910–1931
(1923–1926 = 100)

Year	Barite	Feldspar	Fluorspar	Fuller's Earth	Phosphate Rock	Sulfur	Salt	Talc
1910	27	12	67	27	69	15	60	59
1911	25	46	71	33	79	15	61	58
1912	27	43	85	29	77	18	66	68
1913	35	61	83	32	76	19	68	84
1914	33	68	63	36	77	20	69	86
1915	47	53	86	37	69	18	75	92
1916	95	66	101	47	76	46	89	104
1917	88	71	139	50	105	67	98	103
1918	66	50	165	56	103	76	102	100
1919	89	36	87	66	83	41	97	86
1920	108	76	126	82	132	91	96	105
1921	33	51	25	64	58	57	70	59
1922	76	66	104	83	74	81	95	99
1923	98	81	98	85	96	97	100	98
1924	93	115	105	99	90	92	96	99
1925	109	104	97	116	114	111	104	103
1926	123	118	122	131	108	124	104	104
1927	138	113	110	144	100	124	106	110
1928	141	118	112	154	116	125	114	116
1929	154	111	120	168	117	146	120	125
1930	122	96	96	182	119	119	113	104
1931	105	82	44	157	70	82	104	95

in 1931 and ½ billion dollars in 1932. In 1932, because of the collapse of expenditures by local governing bodies, even public building contracts were cut in half.

According to their behavior during the recent depression, the building materials can be roughly divided into two groups. One group—including cement, sand and gravel, and crushed stone—did not follow the general break in 1928 and exhibited strong resistance even after its downturn was apparent in 1930. This condition was due mainly to the sustaining influence of increased highway construction during this period, but probably it represents also a continued tendency to use concrete in lieu of other types of construction materials. Dimension stone, likewise, may be considered as more or less in this first group, inasmuch as the production index after following closely that of common brick in earlier years resisted the decline suffered by the brick industry after 1926. The second group comprises materials whose consumption indexes failed even to keep pace with either the volume of general industrial producton or the dollar values of contract awards. The downturn for brick, gypsum, and slate began earlier and proceeded further than the usual thermometers of building activity would indicate.

Fig. 1, plotted from the data given in Table 1, brings out domestic consumption trends for specified building materials.

Chemicals, Fertilizers, and General Industrial Materials.—Domestic consumption indexes for certain important nonmetallic mineral materials other than those used for construction are presented in Table 2. Most of these indexes rise and fall very closely with general industrial production.

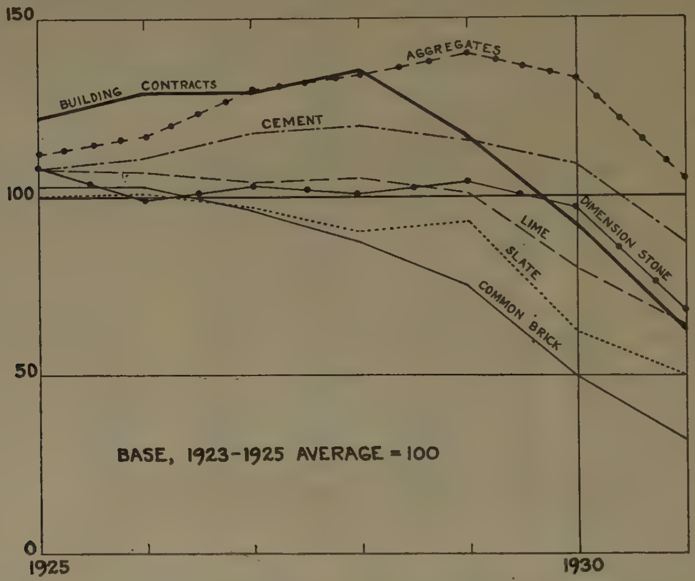

Fig. 1.—Consumption indexes of building materials, 1925–1931.

Salt, because of its diverse uses, is one of the most stable of this miscellaneous group.

Price History

Outside influences, as expressed in the major cyclical movements of general business activity, affect prices as well as production trends. There is a vicious circle wherein production of all classes of goods is limited by purchasing power and purchasing power by production. Demand, to be effective, must be accompanied by purchasing power which, of course, usually depends upon conditions originating completely outside the visible limits of any single industry. To understand price movements in the nonmetallic mineral industries, therefore, we must consider the course of wholesale prices of all commodities.

It would be helpful also to range the nonmetallic minerals themselves into allied groups. This can be done rather easily with respect to building materials (Table 3), but for the purposes of the present paper we are not prepared to classify definitely the other items. Data in terms of index numbers, covering prices of specified minerals that may be described as chemical, fertilizer, or general industrial raw materials, are given in Table 4. For several reasons—in some instances monopoly control, in others foreign competition, in still others a surplus of domestic productive capacity, and above all the wide diversity in conditions of production or consumption—no general correlation is readily worked out, but the figures for these different industries may be used for individual comparisons with other industries.

TABLE 3.—*Price Indexes (Unit Value, Point of Shipment) for Specified Building Materials,* 1910–1931
(1926 = 100)

Year	All Commodities[a]	Building Materials[a]	Cement[b]	Lime[b]	Crushed Stone[b]	Sand and Gravel[b]	Gypsum, Crude[b]	Gypsum Calcined[b]	Common Brick[a,c]
1910	70.4	55.3	52	44	56	49	60	34	[d]
1911	64.9	55.3	49	44	56	52	58	33	[d]
1912	69.1	55.9	47	43	56	56	53	31	[d]
1913	69.8	56.7	59	45	57	49	58	31	40
1914	68.1	52.7	54	43	57	49	56	34	34
1915	69.5	53.5	50	44	56	49	52	33	37
1916	85.5	67.6	64	50	58	54	53	36	49
1917	117.5	88.2	79	69	68	75	66	54	54
1918	131.3	98.6	94	92	90	100	96	70	73
1919	138.6	115.6	100	97	102	107	112	81	97
1920	154.4	150.1	118	116	119	131	142	105	133
1921	97.6	97.4	110	108	113	116	131	108	93
1922	96.7	97.3	103	100	103	111	122	98	105
1923	100.6	108.7	111	108	103	107	117	95	120
1924	98.1	102.3	106	107	102	102	106	102	104
1925	103.5	101.7	104	102	100	103	107	99	89
1926	100.0	100.0	100	100	100	100	100	100	100
1927	95.4	94.7	95	96	97	97	96	92	84
1928	96.7	94.1	92	90	97	93	75	75	79
1929	95.3	95.4	87	86	96	98	78	79	65
1930	86.4	89.9	84	83	95	95	76	104	61
1931	73.0	79.2	65	76	91	93	80	110	61

[a] U. S. Bureau of Labor Statistics.
[b] Calculated from reports of producers to U. S. Bureau of Mines.
[c] Wholesale, New York.
[d] Figures not available.

Table 3 gives price indexes for specified building materials along with the wholesale price indexes for all commodities and for the building materials group as calculated by the United States Bureau of Labor Statistics, and to show the relationship more clearly the figures for certain of the items are plotted in Fig. 2. In all major movements the indexes for lime follow closely those for cement, and those for sand and gravel and crushed stone also have much the same pattern. In 1931, however, the cement index dropped lower even than lime. The preliminary warnings obviously appeared in the building construction industries in 1928—considerably earlier than the depression was manifested in other lines. Whereas both lime and cement prices continued downward after that year, resistance levels were encountered for crushed stone and sand and gravel. These resistance levels for the latter commodities may indicate that prices were previously so low as to leave little or no profit margin even for plants that were relatively economical. The gypsum curve is likewise interesting. In 1920, the price rose much faster than the prices of the other items listed, approaching the record made by the building materials group as a whole, and thenceforth it yielded more gradually to the downward pressure, until in 1928 it broke rather sharply. That this break was excessive is indicated by the subsequent upward trend which brought crude gypsum in line with other items in 1931. The figures for calcined gypsum sales, which of course are more closely correlated than crude gypsum sales (mostly to cement plants) with building construction, exhibited an even more drastic reversal in trend, beginning in 1930.

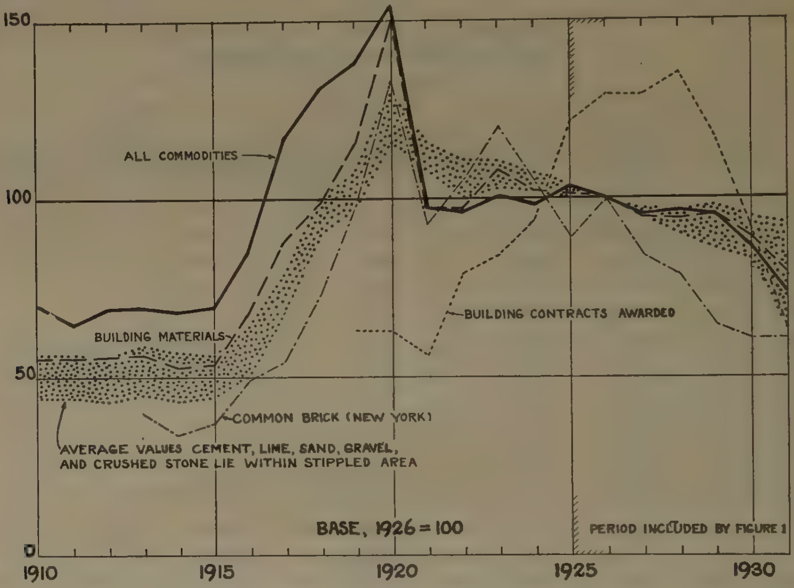

FIG. 2.—PRICE INDEXES OF BUILDING MATERIALS, 1910–1931.

Data of the character shown in Tables 3 and 4 can be made the subject of much fruitful study. For those who care to pursue this matter further, it may be well to mention that the apparent spread between building material prices and general commodity prices during and after

TABLE 4.—*Price Indexes* (*Unit Values, Mine Shipping Point*) *for Specified Chemical Fertilizer and General Industrial Raw Materials,* 1910 *to* 1931
(1926 = 100)

Year	All Commodities[a]	Chemicals[b]	Barite	Feldspar	Fluorspar	Fuller's earth	Phosphate rock	Sulfur	Salt	Talc
1910	70.4		38	44	48	62	121	100	55	82
1911	64.9		43	41	39	66	115	101	56	74
1912	69.1		55	44	36	65	116	96	59	74
1913	69.8	80	47	43	35	67	111	98	62	75
1914	68.1	81	40	40	32	69	104	101	61	77
1915	69.5	112	47	40	31	71	87	94	64	74
1916	85.5	161	62	45	33	73	88	88	63	76
1917	117.5	165	76	48	57	74	89	119	84	81
1918	131.3	182	91	64	114	94	97	122	109	94
1919	138.6	157	112	72	140	131	150	84	116	94
1920	154.4	165	126	82	139	136	181	110	128	107
1921	97.6	115	108	88	113	130	176	99	145	97
1922	96.7	100	98	94	98	115	128	91	119	106
1923	100.6	101	105	95	114	105	114	89	115	102
1924	98.1	99	106	96	108	103	105	90	111	109
1925	103.5	102	101	92	99	99	98	88	104	95
1926	100.0	100	100	100	100	100	100	100	100	100
1927	95.4	97	89	92	99	99	105	103	96	101
1928	96.7	96	88	88	104	95	105	100	97	110
1929	95.3	94	90	84	105	95	103	100	94	103
1930	86.4	89	88	81	100	90	105	100	91	101
1931	73.0	79	77	76	96	74	108	100	86	99

[a] U. S. Bureau of Labor Statistics.

[b] Chemicals and drugs, combined index, U. S. Bureau of Labor Statistics.

the World War is explained in part by the statistical method employed; if the data are calculated on a 1913 base a somewhat different picture is presented. It may also be noted that the United States Bureau of Labor Statistics indexes are based on quoted prices—mostly at consuming points and hence including freight—whereas those presented for the first time in this paper are annual averages (f.o.b. point of origin) calculated from reports of producers to the Bureau of Mines. The latter are more accurate in so far as they more closely represent returns from actual sales, but they are subject to criticism in that they do not reflect minor geographical shifts in the centers of gravity of production. Only those industries are chosen which supply the bulk of the domestic demand for their product, thereby minimizing foreign influences, but no correction is attempted for variations in local markets. In 1929, for example, the average domestic price of cement was $1.48 per barrel, f.o.b. factory, but this is a composite which includes a range from $1.13 per barrel in Alabama to over $2 a barrel in the Rocky Mountain States.

Variations in grades also exist, but it is doubtful whether these inaccuracies are of sufficient magnitude to exert a pronounced effect on the indicated trends.

Size of Plants

The average size of individual operations has increased greatly in recent years—another sign that the nonmetallic mineral industries are growing up. This trend is a modern tendency in various lines of industry and follows from the fact that there seems to be in all industries an optimum size of individual operations. In the heavy building materials industries, the most economical size may be determined to a greater extent by the magnitude of the local market than by potential savings in operating costs. That a nice balance exists between the economics of mass production and the disadvantages of higher transportation charges is clearly shown by the increasing number of portable sand and gravel and crushed stone plants—an indication that in these industries centralization had been overdone. The economics of this reversal in trend has been discussed by Thoenen.[2]

The general increase in the average size of plants has come not alone from expansion of larger operations but to at least an equal degree from the elimination of very small units. It is possible to prove even by statistical methods that these small units are not economical. In Table 5, data for which were calculated from reports of the Federal Census, it is shown that the average value of products per mine or quarry has increased much more than the average number of employees. Owing to increased mechanization and improved general efficiency, the value of output per wage earner was in several cases about twice as large in 1929 as it had been in 1919, and as prices were generally lower in 1929 than they were in 1919, the increase in volume of product per wage earner was doubtless even more striking. Inasmuch as the Census of 1929 included all plants producing products valued at over $2500 (instead of $5000 in previous Censuses), the figures for 1929 may include in some cases a larger number of small plants, thereby tending actually to reduce the apparent magnitude of the trend.

Despite this recent growth the nonmetallic mineral-producing industries are largely composed of relatively small units. In 1929, the value of the average output of copper was $1,575,000 per mine; for iron ore the average output per mine was valued at $948,000; for lead and zinc it was $300,000 and even for bituminous coal it was $171,500. Table 5 shows that only five nonmetallic mineral mining industries reported a production value averaging over $100,000 per plant; namely, basalt (trap rock),

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[2] J. R. Thoenen: Prospecting and Exploration for Sand and Gravel. U. S. Bur. Mines *Inf. Circ.* 6668 (1932) 2–3 and 10–12.

TABLE 5.—*Changes in Average Size of Individual Operations*, 1909–1929
(Calculated from Federal Census Data)

Industry	Wage Earners per Mine or Quarry			Value of Products per Mine or Quarry (Thousands of Dollars)		
	1909	1919	1929	1909	1919	1929
Limestone[a]	15.7	23.8	25.7	15.6	57.2	93.4
Granite	22.7	21.1	23.1	22.9	47.9	69.8
Basalt	23.9	11.2	21.0	25.4	55.7	108.0
Slate	40.3	33.7	31.5	27.7	55.0	80.6
Marble	57.1	27.9	38.0	57.8	71.0	85.7
Sandstone[b]	7.5	15.5	12.5	7.1	38.7	36.7
Miscellaneous stone[c]			7.9			36.2
Abrasive materials	7.72	9.32	12.8	8.51	21.2	39.2
Asbestos	2.7	13.3	17.7	3.3	22.7	36.1
Barite	5.7	9.4	19.2	5.4	16.2	40.9
Clay	9.7	15.6	17.5	8.78	28.7	45.6
Feldspar	8.8	10.9	10.3	9.79	18.3	33.4
Fluorspar	19.3	15.6	29.3	19.2	46.3	79.2
Fuller's earth	15.6	91.5	41.3	15.0	224.3	201.0
Gypsum	15.6	45.7	32.9	26.2	142.0	91.0
Magnesite	3.8	40.7	70.2	5.32	197.0	40.8
Mica	3.5	6.5	7.1	2.6	8.8	16.1
Mill and pulpstones	3.6	3.4	11.7	2.46	5.9	44.4
Phosphate rock	51.5	63.4	96.8	70.5	149.0	394.0
Silica[d]	11.3	5.7	19.6	16.5	12.8	63.5
Sulfur and pyrites	90.3	105.0	219.9	320.0	924.0	3,713.0
Talc and soapstone	27.3	31.9	19.6	25.5	76.6	96.0
Miscellaneous nonmetals[e]	9.7	12.8	16.1	10.7	28.7	184.0

[a] Data for the quarrying of limestone by cement and lime producers are included in the statistics for 1929, but are not included in those for 1919 and 1909.

[b] Including for 1919 and 1909, but excluding for 1929, data for quarrying enterprises whose final product (glass sand, molding sand, or silica material) was derived from sandstone, siliceous mica schist (for use as ganister), etc. The data for these enterprises are included in the glass sand, molding sand, or silica industries for 1929.

[c] Included in figures for the other stone industries (principally basalt and sandstone) for 1919 and 1909.

[d] In 1929 includes diatomite, 10 enterprises; ganister, 18; quartz, 9; quartzite, 2; silica rock, 6; silica sand, 14; siliceous mica schist, 3; tripoli, 8.

[e] In 1929 includes borates, 2 enterprises; cyanite, 2; graphite, 5; lithium minerals, 4; mineral pigments, 4; tantalum, 1; vermiculite, 1.

fuller's earth, phosphate rock, sulfur (included in the statistics), and "miscellaneous" (which includes borates). Really large plants producing as much as $1,000,000 worth of products are found only in the limestone (10), sand and gravel (8), granite (4), basalt (1), miscellaneous stone (1), and a very few others such as magnesite, sulfur, and borates, for which Census data do not reveal separate figures.

TABLE 6.—*Size of Individual Enterprises*, 1929
(Data from Federal Census)

Industry	Number of Enterprises Producing Products Valued at					
	Less than $20,000	$20,000 to $49,999	$50,000 to $99,999	$100,000 to $249,999	$250,000 to $499,999	$500,000 or over
Limestone	407	259	183	219	71	28
Sand and gravel	71	352	263	206	42	23
Granite	196	69	61	54	15	11
Basalt	34	33	19	36	10	5
Clay	91	53	29	16	8	2
Slate	41	23	23	22	10	1
Stone—miscellaneous	126	46	19	7	3	3
Marble	20	14	16	13	4	3
Sandstone	68	41	20	13	3	0
Phosphate rock	2	3	3	4	7	7
Gypsum	14	10	11	21	3	1

Data showing the scale of operations per enterprise (including output from one or more plants operated by the same concern) in 1929 for industries for which separate figures are available are summarized in Table 6.

An examination of the Census figures indicates that in most of the industries listed in Table 6 the bulk of the domestic output is furnished by plants of medium size, the $100,000 to $250,000 group contributing usually at least 25 per cent and often a larger portion of the total.

This statistical study might be extended to cover cement, clay products, and glass, but as elsewhere in this paper the purpose is mainly to indicate a procedure for more detailed economic analysis.

TYPE OF OWNERSHIP

The data in Table 7 as to type of ownership are abstracted from the Census of Mines and Quarries (1929 General Report, p. 11). For the mining industries as a whole, corporations operated 66 per cent of the total number of mines and quarries, employed 94.7 per cent of the wage earners, and produced 95.7 per cent of the total value of the products. "Other" forms of organization include partnerships, enterprises operated by individuals, cooperative enterprises, and so forth.

ORGANIZATION OF THE INDUSTRIES

Because of the diversity of products represented, the nonmetallic mineral industries are not organized as a group, and the small size of most of the individual industries has tended to delay the growth of industry consciousness and even to prevent the carrying on of active research either in the plants or in the fields of marketing. Portland cement manufacture and the clay products industries are the only ones that can

TABLE 7.—*Type of Ownership*, 1929
(Data from Federal Census)

Industry and Type of Ownership	Number of Enterprises	Number of Mines and Quarries	Wage Earners (Average for Year)	Value of Products
Limestone: total	1,167	1,256	32,300	$117,257,784
Corporate	746	805	27,692	104,969,760
Other	421	451	4,608	12,288,024
Sand and gravel: total	957	1,165	15,994	102,311,914
Corporate	738	913	14,147	91,673,784
Other	219	252	1,847	10,638,130
Granite: total	406	434	10,037	30,381,373
Corporate	203	223	8,371	25,901,847
Other	203	211	1,666	4,479,526
Basalt: total	137	144	3,053	15,543,687
Corporate	89	96	2,620	13,786,397
Other	48	48	433	1,757,290
Clay: total	199	236	4,139	10,753,445
Corporate	122	136	3,349	8,392,460
Other	77	100	790	2,360,985
Slate, total	120	130	4,098	10,486,390
Corporate	70	74	3,374	8,850,940
Other	50	56	724	1,635,450
Stone, miscellaneous: total	204	234	1,841	8,475,008
Corporate	60	87	971	5,948,302
Other	144	147	870	2,526,706
Marble: total	70	88	3,350	7,538,905
Corporate	61	79	3,301	7,380,787
Other	9	9	49	158,118
Sandstone: total	145	172	2,156	6,311,977
Corporate	74	90	1,574	4,642,894
Other	71	82	582	1,669,083

boast an annual output valued at substantially more than 100 millions of dollars annually—although the annual output of the sand and gravel industries frequently rises slightly above this figure, and the stone industries likewise may be placed in this category by lumping crushed stone and the diverse kinds of dimension stone into a single statistical unit. Glass manufacture would also be in this small list of large industries were it not for the fact that the production figures have usually not been included with those for other nonmetallic mineral industries. Glass is in many respects as strictly a nonmetallic mineral product as common brick or even cement, but from the practical viewpoint of making statistical canvasses, the glass industry is not considered as a branch of the mining industry but only as a purchaser of its product. A second group —comprising sulfur, gypsum (including certain gypsum manufactures), lime, salt, and phosphate rock—may be classed as moderately large, the output of each of these products normally exceeding 10 million dollars annually. The industries producing the other 20 or 30 nonmetallic minerals are individually small, and only one or two have succeeded in maintaining any even semipermanent cooperative trade organizations. Trade associations, however, do exist in most of the really large industries and have contributed greatly to the rapid advancement and general improvement of conditions in every instance where a reasonable degree of cooperative support has been achieved.

Financial consolidations have been of importance likewise in only the larger industries. Most of these combinations have been horizontal in type, ownership of plants in several strategic positions representing a real economic advantage in the case of bulky products, and more especially those which like gypsum or cement may be given nationally advertised brand names. There are a few examples of vertical consolidations but not enough to indicate a general trend in this direction. The marketing of ready mixed concrete has been undertaken recently by a number of

TABLE 8.—*Miscellaneous Data*

(Calculated from Federal Census Data)

	Ratio of Wage Earners to Salaried Employees[a]	Value of Output Per Wage Earner	Percentage of Total Value of Products Expended for			
			Salaries[a]	Wages	Supplies and Materials	Fuel and Purchased Electric Energy
All mines and quarries.....	15.2	$2,960	5.3	46.7	12.3	4.9
Coal (total)...............	19.2	2,250	5.0	59.5	11.1	3.9
Copper....................	12.6	6,370	3.5	25.8	15.5	5.4
Iron ore..................	13.0	6,950	2.6	20.8	9.4	5.0
Limestone.................	9.8	3,620	6.1	33.5	17.8	7.2
Sand and gravel...........	4.1	6,400	10.5	22.3	11.7	8.7
Granite...................	8.7	3,030	8.5	41.6	11.9	5.8
Clay......................	9.7	2,610	7.3	34.9	13.9	7.1
Slate.....................	11.6	2,560	6.8	46.5	6.3	6.0
Marble....................	13.7	2,250	7.7	43.7	7.5	7.2
Phosphate rock............	11.1	4,070	5.4	25.3	11.8	15.2
Basalt....................	8.9	5,090	5.9	29.0	14.2	6.7
Sulfur and pyrite.........	7.2	16,850	2.6	9.4	9.0	8.3
Sandstone.................	7.0	2,930	8.4	41.7	15.6	6.3
Stone—miscellaneous.......	4.6	4,600	6.7	28.3	19.2	5.3
Gypsum....................	15.3	2,770	5.3	45.7	13.8	7.3
Silica....................	6.4	3,240	11.3	36.1	13.2	6.1
Sand molding..............	4.7	4,610	12.8	27.1	7.4	8.5
Fluorspar.................	8.0	2,710	9.8	39.0	21.9	6.2
Glass sand................	10.9	5,200	5.2	24.4		
Fuller's earth............	9.4	4,860	6.3	17.7	8.8	9.3

[a] Including proprietors and firm members, corporation officers, clerks, etc., but not employees of central administrative offices (if any).

producers of aggregate and since time immemorial certain industries —notably brickmaking—have been highly integrated. A few glass companies have control of glass-sand deposits, and petroleum companies generally produce most of the fuller's earth they require. Some of the paint manufacturers mine, as well as purchase, some of their raw materials. The pottery industry, on the other hand, purchases its materials almost exclusively and even the firms that cater to this industry generally confine their operations to one class of products such as clay or feldspar instead of including a diverse class of mining operations under the same management. Some diversification follows as a natural result of the production of such by-products as mica from clay or feldspar mining, and the affiliation of a large cement manufacturing organization with a steel company owes its origin to the employment of blast-furnace slag as a raw material for cement making. Such isolated developments, however, are not indicative of general tendencies.

A wealth of data with regard to trends in production, ratios of various production expenditures to value of output, market flow, etc., are available in the publications of the Census Bureau. Table 8 indicates how certain of these data may be tabulated for comparison both with similar data for previous years and as between different industries. Comparisons between plants in various states can also be made in this manner, so as to indicate the relative importance of low wages or cheap power as reflected in actual costs. The Census data, it should be remembered, do not permit of calculating total costs, but properly interpreted they do give an excellent basis for analyzing the main elements of operating costs.

ACKNOWLEDGMENTS

In this discussion general theories have been drawn upon without an attempt to trace back through various economic textbooks to discover the original references. The manner of treatment is believed to be new, and, it is hoped, suggestive. The authors gratefully acknowledge the helpful advice of Mr. F. E. Berquist, formerly statistician for Mines and Quarries, Bureau of the Census, now of the Bureau of Mines; and of the Messrs. Paul Hatmaker, and H. Herbert Hughes, both of the Building Materials Section, U. S. Bureau of Mines, who also prepared the charts.

DISCUSSION

(*H. Foster Bain presiding*)

H. F. Bain,* New York, N. Y.—My observations indicate a growth in small plants, due largely to the tendency for men who are thrown out of work to strike out on their own.

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* Managing Director, Copper & Brass Research Association.

The average cost of about 160,000 homes built in 1931 was around $8000; whereas the A. O. Smith Co. is now figuring on the construction of $2500 homes in quantities.

C. M. SMITH,* Urbana, Ill.—Within the past three years there has been a marked increase in the number of small nonstripping coal mines in Illinois.

P. M. TYLER.—The most economical size of industrial plant is one having an annual output valued at from $100,000 to $200,000 based on 1929 prices. The optimum size of plant is predicated largely on its local market.

H. F. BAIN.—There seems to be a disparity between the average low yield gold sluicers expect from their gravel and the high prices that building gravel commands.

P. M. TYLER.—In Denver some sellers of building gravel are sluicing the ground preparatory to marketing it for construction purposes. As to the high price, building gravel has to meet rigid specifications, and hence must be well prepared. In general, the management of gravel plants is alert, competition is keen and it is unlikely that profits are large, as there is much overdevelopment there as elsewhere.

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* Research Associate, Department of Mining Engineering, University of Illinois.

Milling Methods and Costs at the No. 2 Concentrator of the Phosphate Recovery Corporation

By H. S. Martin,* Mulberry, Florida

(New York Meeting, February, 1933)

The Phosphate Recovery Corporation operates three flotation plants, Nos. 1 and 2 concentrators about three miles northeast of Mulberry, Florida, and No. 3 plant at Wales, Tennessee. These plants represent the first commercial phosphate flotation operation and are also believed to be the largest nonmetallic flotation operation in the world at the present time.

The details of this paper will be confined to plant No. 2, this being the newest and largest concentrator in Florida, and the flow sheet being practically the same as at plant No. 1. The No. 3 plant has been operated but a short time and has been shut down since December, 1932, on account of the decrease in phosphate rock production.

The phosphate washers usually are located centrally on a known deposit so that about two hundred acres of ground can be mined economically to each washer location. The washers are moved to new areas when the distance becomes too great for economical pumping. The flotation plants are built either adjacent to a washer, so that current debris may be utilized, or on the site of a former washer where there is sufficient accumulated debris available for mining, and the flotation plants must also be moved from time to time.

Most of the water used in the Florida flotation plants is recovered after passing through a circulating system where most of the fine suspended matter is settled out, but some make-up water is obtained from deep wells near the plant.

Power is obtained from the Tampa Electric Co. through the distribution facilities of the International Agricultural Corporation.

Flotation concentrate is hauled from the plant by trolley over the International Agricultural Corporation's trolley system, to the drying plant at Prairie, one mile northwest of Mulberry.

Ore Treated

In the pebble phosphate district of Florida, the phosphate mineral, or collophane, is usually found in horizontal beds of varying thickness covered by varying depths of overburden. At present the overburden is usually removed by electric dragline and the actual mining of the phosphate matrix is done by hydraulic means. The phosphate pebbles and grains usually are embedded in a matrix of sand and clay and the methods of washing and screening have been described frequently.

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* Manager, Phosphate Recovery Corporation.

The older washing methods are briefly described as follows: Hydraulic guns using water at 200 lb. per square inch cut into and disintegrate the "matrix," which is then carried by the water to a pump sump, whence it is pumped by 10-in. or 12-in. sand pumps to the washer. The pump discharge passes first through a trommel screen, and all plus 1½-in. material, mud balls and rock, are rejected. Undersize from the trommel usually passes over a flat screen of slotted metal, with openings ³⁄₃₂ to ³⁄₆₄ in. wide. Undersize from the flat screen is washer tailing or "debris." The partly dewatered oversize from the flat screen usually passes over vibrating screens and thence through a series of two spirally bladed "log-washers" that break up remaining lumps of stiff clay. Overflow from both log-washers is also waste. The rake discharge from the second log-washer passes over vibrating screens, where it is separated into two sizes of finished product, fine and coarse pebble. A more complete description of this part of the process was given in *Pit and Quarry* for Dec. 30, 1931.

After the clay is worked into suspension the washer process is one of size separation, most of the phosphate grains being larger than the quartz grains with which they are associated. In every deposit, however, there is more or less phosphate rock of approximately the same size as many of the sand grains, and size separation for this material is impossible. The specific gravity of phosphate particles and quartz particles is so nearly the same that gravity concentration methods are not feasible. All such material was formerly wasted by the washers, and was called "debris." It is this "debris" that furnishes feed for the flotation plants.

The amount of rock formerly wasted varies in different localities, but in a great many instances represents a tonnage equal to the tonnage of "pebble" originally obtained by the washer. Concentration of this finer material therefore will almost double the tonnage of output from a given area of matrix. It is also true that the "coarse pebble" usually contains inclusions of quartz, clay and feldspar and is, consequently, of lower grade than the flotation concentrate made from the finer portions of the same matrix, particularly if the flotation feed is ground.

Feed for the No. 2 plant is obtained by remining several old debris ponds. Most of the debris consists of quartz and phosphate grains ranging in size from minus 8 mesh to plus 200 mesh, although there is some material larger than 8 mesh and some smaller than 200 mesh. At the time the debris was discharged by the old washers a great deal of clay was carried away from the sandy material by the water and deposited at a distance farther away from the washer site. Hydraulic mining is used in handling this debris, although the water pressure here is only 50 to 60 lb. per square inch. At the No. 1 plant current washer debris from the No. 11 washer of the International Agricultural Corporation is used as feed for flotation and is pumped as produced to the flotation feed bins.

Although at flotation size (minus 28 to minus 35 mesh) there are very few apparent inclusions of quartz in the phosphate particles, the chemical composition of the rock seems to vary in different localities. Apparently clean phosphate particles from one deposit may assay as low as 68 to 70 per cent $Ca_3(PO_4)_2$ (bone phosphate of lime) while similar grains from another locality may assay as high as 78 to 80 per cent bone phosphate of lime. Collophane, which is the predominant mineral in these deposits, is generally considered to be a complex calcium carbono-fluo-phosphate, the fluorine and carbon dioxide being tied up molecularly with the phosphate. Separation of these constituents is impossible, of course, by flotation, the main object in flotation being to separate the collophane particles from the quartz particles.

TABLE. 1.—*Screening Analysis of Crude Debris as It Comes from Pit*
PHOSPHATE RECOVERY CORPORATION, PLANT NO. 2

Mesh	Weight, Per Cent	Weight, Cum, Per Cent	Bone Phosphate of Lime, Per Cent
Original	100.00		24.79
+ 10	2.58	2.58	55.44
+ 14	1.41	3.99	73.62
+ 20	3.75	7.74	73.52
+ 28	8.43	16.17	54.64
+ 35	19.20	35.37	27.02
+ 48	23.65	59.02	16.77
+ 65	21.31	80.33	12.76
+100	10.77	91.10	13.09
+150	4.45	95.55	9.44
−150	1.17	96.72	26.52
Slime ([a])	3.28	100.00	28.42

[a] Sample stirred in pail with water at 15 to 20 per cent solids, allowed to settle 10 seconds, then decanted. Operation repeated three times.

In a rather complete investigation made in 1929 by Prof. R. J. Colony and Paul F. Kerr for the International Agricultural Corporation, their conclusions were in part as follows:

> Comparisons of X-ray diffraction patterns show that the phosphate constituents in the various phosphate rocks (Tennessee and Florida) are apparently in the form of a complex mineral of the apatite type, probably staffelite. This mineral contains carbonate and other chemical constituents peculiar to carbono-apatite and although complex, from the chemical standpoint, the mineral staffelite is present in all types of phosphate rock examined. Although shown by X-ray examination to be crystalline,

it is possible that microscopic examinations would show it to be so finely grained as to appear a solid jelly-like mass devoid of crystalline character.

This description of the phosphate rock probably explains why the finer portions, commonly known as "slimes," are so difficult to treat by flotation. This is true particularly of Florida pebble phosphate. There are many varieties of more coarsely crystalline apatite that do not form a near colloidal "slime" when ground.

Table 1 gives a complete screening analysis of a typical sample of "debris" as it comes from the dump. It requires about one cubic yard of this material to yield one long ton of ground and deslimed flotation feed.

History of Concentrator Operations

Early in 1926, Mr. John T. Burrows, vice president of the International Agricultural Corporation, in charge of phosphate rock operations, instigated an investigation as to possible methods of saving the fine rock then being wasted by the washers. James A. Barr, chief engineer, and Charles E. Heinrichs, of the Rock Department of the Corporation, were detailed to carry out this investigation.

After some experimental flotation work, it was discovered that Minerals Separation North American Corporation already owned at least one patent[1] covering the concentration of phosphate rock by flotation. That corporation was accordingly consulted and an agreement reached whereby the two companies would join forces in developing a commercial method of flotation concentration. This resulted in the formation of the Phosphate Recovery Corporation.

After a great deal of experimental and research work in the laboratories of both the Minerals Separation North American Corporation and the International Agricultural Corporation, a pilot plant was built in 1927 adjacent to the No. 11 washer of the last-named company, where current washer debris was available for experimental work. Many mechanical difficulties were encountered and a great many expensive changes made in equipment and finally a fairly satisfactory method worked out. During this time and up to the present, an active research program was carried out in the Phosphate Recovery Corporation research laboratory in Mulberry with a view to lowering costs and bettering metallurgy. At present six men are engaged in research and development work at the laboratory. This work has resulted in many improvements in the method of procedure. Ideas and methods developed by the laboratory have resulted in large savings in the commercial operations.

In January, 1929, the present Phosphate Recovery Corporation plant No. 1 was completed and commenced operations. During the pilot plant operation reagent cost was a considerable item but this was gradually reduced by desliming prior to flotation and the use of fuel oil or other petroleum oil in conjunction with fatty or soap-forming acids. Details of reagent addition also had a considerable bearing on the quantity of reagents necessary. It was found that the reagents for greatest efficiency should be added separately to a thick pulp, and then diluted before actual flotation.

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[1] Broadbridge and Edser: U. S. Patent No. 1547732, July 28, 1925.

After some experience in the operation of Phosphate Recovery Corporation plant No. 1, plans for a new plant having a capacity of about 2440 long tons of feed per day were drawn up and this plant was put into operation in April, 1930.

Plant No. 2, instead of utilizing current washer debris, was supplied with feed from several accumulated debris dumps adjacent to the plant. Although this plant has been in operation only a little over two years, many improvements have also been made in this period. During the early months of operations tailings at this plant averaged 8 to 12 per cent bone phosphate of lime and the grade of concentrate 70 to 72 per cent, whereas in recent months the average tailings have been 4 to 5 per cent B.P.L. and the grade of concentrate 74 to 76 per cent. Most of this improvement has been obtained by preparing the flotation feed so that as little plus 35-mesh material as possible is present. The plant as first designed had no grinding equipment whatever, but eight No. 6 Wilfley tables were installed to treat plus 20-mesh material. This form of table concentration is not gravity concentration, but is really a stratifying classification of suspended phosphate material in the form of floccules. The plus 20-mesh material is treated with reagents just as flotation feed is treated and the concentrate is carried transversely over the riffles while the quartz particles are carried along the riffles by the motion of the tables and discharged at the end of the table. It was found that, although this table separation was efficient if not over one ton per hour was fed to each table, better work might be done on the same feed if it were ground to the proper size and then handled in the regular flotation machines. To test out this method of separation as against the use of tables a 4 by 10-ft. rod mill was installed in May, 1931, and part of the oversize passed through this rod mill and thence to flotation. Much better over-all results and better costs were obtained by the combination of grinding and flotation.

The No. 2 plant, having a capacity of 100 long tons of feed per hour, originally cost nearly $200,000, and it is estimated that the new 60-ton per hour plant can be erected for about $80,000. This new plant cost is then less than $50 per short ton per day capacity as against usual flotation plant costs of $400 to $1000 per ton per day. Cheap plants and low operating costs have been made necessary because of the low value of the finished product and because of the fact that the flotation product must still compete in cost with a pebble product produced by washers.

Fig. 1 gives the present flow sheet of plant 2 and Fig. 2 shows a proposed flow sheet of an improved design of plant of about one-half the capacity of the present No. 2 plant. Changes that would be made in building a new plant are obvious from a comparison of the flow sheets.

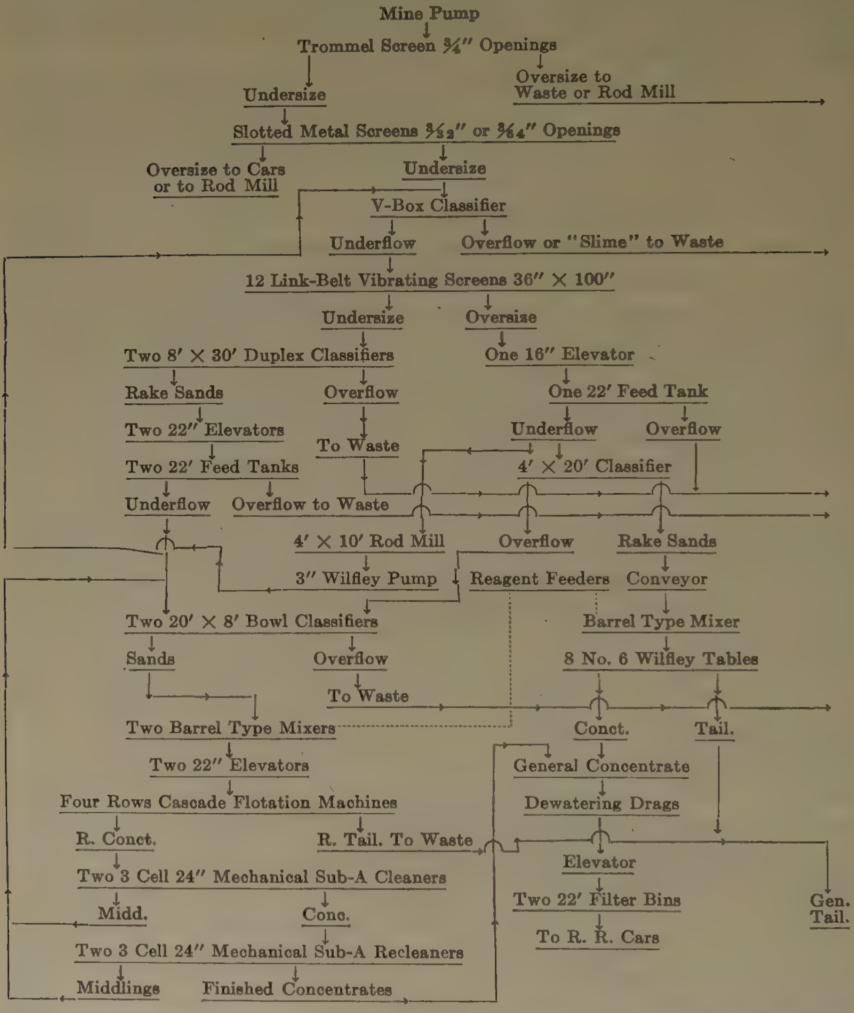

Fig. 1.—Flow sheet of No. 2 Plant, with rod mill, Phosphate Recovery Corporation.

Present Methods of Concentration

At present the debris is pumped at 15 to 20 per cent solids from the pit to a point about 30 ft. above ground level, where it first passes through a revolving trommel screen with ⅞-in. circular openings. The oversize (mostly mud balls) from this trommel may be either rejected or laundered to the rod mill. The undersize from the trommel screen passes over a series of sloping slotted metal screens with openings 1 mm. wide. This screen oversize goes to the rod mill and the screen undersize to the V-box. The V-box is a V-shaped or U-shaped steel tank equipped with an inside overflow launder where part of the slimy water is removed to waste. On the bottom of the V-box are ten 4-in. plug valves through which the partially thickened product is drawn. The V-box underflow passes over twelve 3 by 8 ft. Link-Belt vibrating screens. Rectangular screen cloth of 24 mesh is usually used here. The screen oversize goes to a 16-in. bucket elevator discharging into a 22-ft. diameter cone-bottom steel bin. The vibrating screen undersize goes to two Dorr 8-ft. duplex classifiers, the overflow from these classifiers going to waste and the rake products discharging into two 22-in. bucket elevators that discharge into two 22-ft. diameter cone-bottom steel bins. Each of the

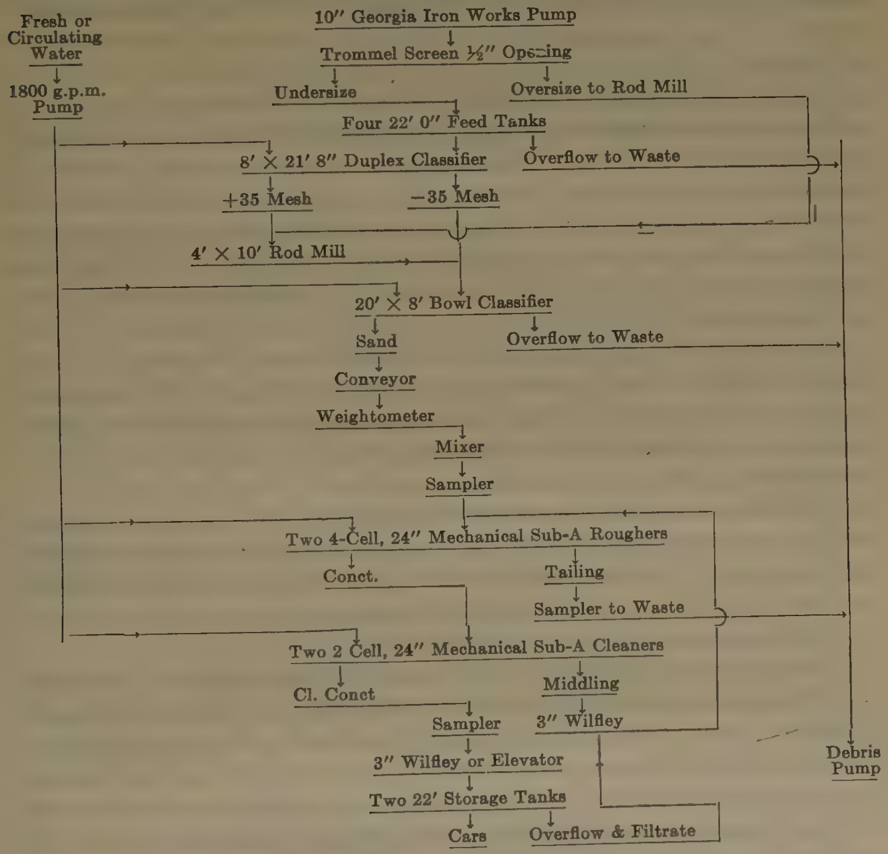

FIG. 2.—PROPOSED NEW FLOW SHEET, 60 LONG TONS FEED PER HOUR.

three bins has a storage capacity of about 300 tons. The duplex classifiers are operated to give approximately a 150-mesh separation, rejecting all material much finer than this, including most of the clay and other very fine suspended material.

The loss of very fine material in the several overflows (Fig. 1) varies greatly with the material treated, but does not amount to any considerable portion of the total phosphate material in the original feed. Investigations are now being carried on with a view to cutting down losses in this respect.

The rock from the coarse-rock bin may be treated in two ways. It may be drawn to the 4 by 10-ft. rod mill, and thence returned by 3-in. Wilfley pump to the V-box in closed circuit with the vibrating screens, or it may be drawn to a 4-ft. Dorr simplex classifier, sands from which are conveyed on an 18-in. belt up to a barrel-type mixer where reagents are added prior to table concentration. Overflow from the simplex classifier is laundered to join the fine bin underflow to two 20 by 8-ft. Dorr bowl classifiers.

At the present time the simplex classifier and table plant are used only when the rod mill has not sufficient capacity to grind all the coarse material to screen size. When an excess of coarse material is being mined, part of the coarse bin discharge goes to the rod mill and part to the simplex classifier and thence to the tables.

The discharge from the two fine-feed storage bins goes direct to 20 by 8-ft. Dorr bowl classifiers where the flotation feed receives a final washing and desliming. The overflow from the two bowls goes to waste and the rake product passes on to two cylindrical reagent mixers, each 16 ft. long by 5 ft. in diameter.

The flotation agents are added to the bowl rake product. Sufficient alkali is used to give a slight alkalinity to the pulp. The other agents are a fatty acid, a collector and a small proportion of a strong frother. Other reagents have been used successfully but the present combination has, to date, given the lowest cost. With the accumulation of experience, the total reagent consumption has been brought lower and lower, until now, on the average feed, it is less than 1½ lb. to the ton of feed.

The reagent mixers are rotating cylinders set at a slope of 3° 40′ and equipped with interior baffles for agitating the feed with the reagents. They discharge into the boots of two 22-in. bucket elevators, which discharge into the cascade flotation machine headers. Dilution water is added to the elevator discharge to give approximately 25 per cent solids in the first cascade cells. In the reagent mixers the solids average 60 to 70 per cent. There are four rows of cascade flotation machines, each row consisting of one 10 by 3-ft. cell and five 8 by 3-ft. cells in series. The rougher concentrate from the cascade machines goes to two three-cell 24-in. mechanical subaeration machines. The primary cleaner concentrate passes on to two more three-cell mechanical subaeration machines for a second cleaning. Cleaner tailings or middlings from both operations are pumped back to the 20-ft. bowl classifiers by a 3-in. and a 4-in. Wilfley pump.

In the light of present knowledge, we believe that mechanical subaeration machines will give better results as roughers than the cascade machines, and mechanical subaeration roughers are, therefore, used in the new design.

The regular mechanical subaeration design was changed somewhat on the machines used in phosphate flotation. Instead of a single spitz, these machines are double spitz and discharge at both sides, weirs being located at the center of the dividing compartment between the cells. This was done because of the low ratio of concentration and the very rapid float obtained on this type of feed. Air for the twelve mechanical subaeration cleaner cells is furnished by an Allen type A centrifugal blower, delivering 600 cu. ft. of air per minute at 1.25 lb. per sq. in. pressure. It is direct connected to a 7½-hp. motor of 3600 r.p.m. speed.

The secondary cleaner concentrate or finished product goes to two chain and blade type dewatering drag classifiers and the rake product from these dewatering drags discharges into one 22-in. bucket elevator. The bucket elevator discharges into either of two concentrate storage bins. These bins are of the same size and style as the feed storage bins except that they are equipped with four vertical filter panels covered with one layer of coarse screen cloth and one layer of fine screen cloth. The water drains readily from the concentrate through these filter panels and the partly dewatered concentrate is loaded through a sliding gate into filter-bottom gondola cars. By the time the cars are delivered at Prairie the average moisture content is about 13 per cent. The cars are weighed on track scales at Prairie and the moisture sample taken by hand as they are dumped. The dry weight of concentrate thus determined is used as the basis of mill tonnage calculations.

The Phosphate Recovery Corporation does not handle the drying of the concentrate; this is done by the International Agricultural Corporation, before shipment, in two 6 by 60-ft. Allis-Chalmers rotary kiln-type, oil-fired driers. Details of this process and equipment are given in the article referred to in *Pit and Quarry*, Dec. 30, 1931.

Waste products, including the rougher tailing from the cascade machines, overflow of the bowl classifiers and the overflow of the duplex classifiers, are pumped to a tailings pond by one 10-in. and one 8-in. Georgia Iron Works sand pump. The coarser material in the waste product settles out close to the pump discharge and the water passes on through a series of mined-out pits and canals until most of the suspended matter is settled out. It is then returned for re-use in the plant. Mined-out debris pits are used for tailings disposal as far as possible.

Conveyors, Elevators and Drives

Only one conveyor is used in the No. 2 plant, delivering table feed from the simplex classifier to the reagent mixer. This belt is 18 in. wide, 115 ft. between pulley centers and is inclined at 14°. It is driven by a 10-hp., 220-volt, 1150-r.p.m. motor, through speed reducer and sprocket and chain drive. Belt speed is 212 ft. per minute.

All elevators are set at 30° from vertical in order to assist dumping, and all pulleys are 59 ft. apart center to center. High-pressure sprays are also used in dumping the elevators as the feed handled has a strong tendency to remain in the buckets. The four 22-in. elevators handling fine feed and oiled feed are equipped with 18 by 10 by 10½-in. buckets, spaced 24-in. centers, and are driven 320 ft. per minute. Drives are 25-hp., 220-volt, 1150-r.p.m. motors, connected through enclosed gear-reduction units and sprocket and chain. The sprocket and chain was used in addition to gear reducers so that belt speed might be more easily changed, because it was known that this material is difficult to dump from buckets. The 22-in. concentrate elevator is the same as the feed elevators, except that it is driven at a slower speed of 164 ft. per minute. Pulleys for the 22-in. belts have 24-in. faces and are 36 in. in diameter. All are lagged with belting to prevent sand wear.

The coarse rock elevator is 16 in. wide, equipped with 14 by 8 by 8-in. buckets, spaced 24-in. centers. It is driven by a 10-hp., 220-volt, 1150-r.p.m. motor, through gear-reduction unit and sprockets at 320 ft. per minute.

The Link-Belt vibrating screens are individually driven by a 2-hp., 220-volt, 1720-r.p.m. motor through V-belt drive. Duplex and bowl classifier rakes are driven at 27 strokes per minute, by 10-hp., 220-volt, 860-r.p.m. motors through V-rope drives. The bowl rakes are driven at 3.25 r.p.m. by 5-hp., 220-volt, 1140-r.p.m. motors through V-rope and worm gear.

The 4 by 10-ft. rod mill is driven at 28 r.p.m. by a 75-hp., 2300-volt, 870-r.p.m. motor. The motor is connected by 12 V-ropes to the pinion shaft.

The 5 by 16-ft. flotation reagent mixers are driven at 14 to 16 r.p.m. by 15-hp., 220-volt, 1150-r.p.m. motors through V-rope, sprocket and chain and pinion drives to a ring gear. They are mounted on rollers at a slope of 3° 40′ and were especially designed for this purpose. The table feed reagent mixer is 30 in. in diameter by 16 ft. long and is similarly mounted and driven by a 7½-hp., 220-volt, 1140-r.p.m. motor at 16 r.p.m., through speed reducer and sprocket and chain.

Each of the No. 6 Wilfley tables is driven by a 1½-hp., 220-volt, 1165-r.p.m. motor through V-ropes.

The mechanical subaeration flotation machines are divided into two six-cell gear-type units, each driven by a 40-hp., 220-volt, 860-r.p.m. motor, through V-ropes, impeller speed being 345 r.p.m.

The concentrate dewatering drags are driven by 5-hp., 220-volt, 1140-r.p.m. motors through sprocket and chain and gear reducers.

The 4-in. Wilfley middling pump is driven by a 30-hp., 220-volt, 1165-r.p.m. motor, direct connected and the 3-in. Wilfley middling pump by a 15-hp., 220-volt, 1160-r.p.m. motor also direct connected.

The 3-in. Wilfley pump used to return rod-mill discharge to the V-box is driven by a 15-hp., 220-volt, 1160-r.p.m. motor, direct connected.

The 8-in. Georgia Iron Works tailings pump is driven by a 100-hp., 2300-volt variable-speed slip-ring motor direct connected, and the 10-in. Georgia Iron Works tailings pump by a 150-hp., 2300-volt variable-speed slip-ring motor, also direct connected.

The entire mill-water supply is furnished by one low-pressure and one high-pressure pump. The low-pressure pump is 12 by 10 size, delivers 5000 gal. per min. at 100-ft. head, and is direct connected to a 150-hp., 2300-volt, 1165-r.p.m. motor. The high-pressure pump is 3 by 2½ size, delivers 225 gal. per min. at 230 ft. head and is direct connected to a 30-hp., 220-volt, 1740-r.p.m. motor.

Flotation reagents are stored in tanks at ground level and are pumped to feeders at about 30 ft. elevation by a four-cylinder plunger pump. Feeders are of the cup and disk type, with a swinging adjustable drip tray. Adjustments are made by varying the number of cups and moving the drip tray up or down. The feeders are driven by 1-hp. motors through worm gear-reduction units.

SAMPLING

No adequate provision for automatic samples was made in the original design of the plant, but a Geary-Jennings automatic electric sampler has since been installed to sample the general mill tailing. This cutter discharges into a duck "cradle" where most of the water drains off between cuts.

Concentrate is sampled by a water-dump cutter as it is delivered to the filter bins. Feed samples and intermediate samples of table and flotation products are taken by hand at 30-min. intervals. All samples are collected, dried in an electric oven and prepared for assay at the plant at the end of each 8-hr. shift.

All analytical work is done by the International Agricultural Corporation analytical laboratory.

WATER SUPPLY

As already stated, most of the water used in flotation is returned and re-used, and this is augmented by rainfall, since the average annual rainfall here is about 60 in. The International Agricultural Corporation has several deep wells in the vicinity, however, and in these the water rises to about 30 ft. below ground level. This water is pumped into the circulating system as needed. During long periods of drought the use of larger quantities of this water causes increased reagent consumption, because it is "hard." The circulation water is thoroughly softened by the reagents, but with hard water the reagents form a calcium soap. If much deep-well water were used, it would no doubt pay to soften it prior to use in flotation. Total water used amounts to 5225 gal. per min. or about 2.030 gal. per min. per short ton milled per day. This figure is high, but this is due to the fact that a great deal of water is sent to waste in the "desliming" operation, by the duplex and bowl classifiers, a condition not usual in other flotation plants.

Concentrator Recovery

The month of December, 1931, was chosen as typical for metallurgy and costs, but much lower grade of feed has been handled since then, and metallurgical results are also given for July and August, 1932. In spite of further curtailment during these later months, costs per ton of feed were maintained at about the same figure as for December, 1931. If the plant had been operated full time during July and August, 1932, costs could have been reduced to about 11¢ per short ton of feed. The No. 2 plant has frequently handled 57,000 to 60,000 tons of feed per month, but December was chosen as being average. The tonnages shown for December were made in 24 operating days, the plant being shut down on Sundays and three additional days at Christmas time.

In the fiscal year ending June 30, 1931, this plant shipped 222,607 long tons of concentrate and in the following 12 months shipped 123,383 long tons. The reduction was partly due to curtailment and partly to lower grade of feed handled during the second period.

Table 2 shows metallurgical results for the months of December, 1931, July and August, 1932. Table 3 gives other metallurgical data for December, 1931. Table 4 gives actual screening analyses of monthly composite general mill tailings and concentrate, and a calculated feed screen analysis for December, 1931. Mesh recoveries are also shown. This and other screening analyses show that all the feed should be ground to pass 35 mesh and remain on 100 mesh as far as possible. Equipment at the No. 2 plant is not capable of making this fine a separation. The vibrating screens would not have sufficient tonnage capacity if 28 to 35-mesh screens were used, and one 4 by 10-ft. rod mill has not the grinding capacity necessary for this tonnage. In the new plant design, screens are eliminated and the 35-mesh separation made by classification.

The low grade of the finer sizes in the concentrate does not necessarily indicate that the concentrate would be lower if all the feed were ground to this mesh. With very little difference in floatability between the two predominant minerals (quartz and phosphate), the plant operator applies reagents to float all possible coarse phosphate (plus 35-mesh material), and this condition will inevitably float a good deal of 100 to 150-mesh quartz. If all the phosphate grains were minus 35-mesh or smaller, the separation could be made more sharp, even in the 100 and 150-mesh range. It is a decided advantage, however, to keep most of the feed particles in the 48 to 65-mesh range, for at these sizes the reagent consumption is lower and the float much more rapid than with finer material.

TABLE 2.—*Phosphate Recovery Corporation Plant No. 2. Average Monthly Mill Assays and Recoveries*

Product	Long Tons	Bone Phosphate of Lime, Per Cent	Bone Phosphate of Lime, Recovery, Per Cent	Concentrates Ratio
December, 1931				
Feed	52,323.38	21.374	82.348	4.2639
Concentrates	12,271.00	75.049		
Tailing	40,052.38	4.929		
July, 1932				
Feed	43,265.81	18.214	80.537	5.0509
Concentrates	8,566.00	74.093		
Tailing	34,699.81	4.420		
August, 1932				
Feed	50,859.87	17.786	78.552	5.3145
Concentrates	9,570.00	74.250		
Tailing	41,289.87	4.699		

TABLE 3.—*Metallurgical Data. Concentrator No. 2, Phosphate Recovery Corporation*

PERIOD COVERED, DECEMBER, 1931

Total long tons ore treated[a]	52,323.38	Total average tons concentrate per 24 hr.	538.91
Days operated (actual)[b]	22.77	Recovery of bone phosphate of lime	82.348
Hours operated per day	24 (3 shifts)	Ratio of concentration	4.2639
Average tonnage per 24 hr.	2,297.91		
Total tons concentrate	12,271.00		

[a] All tonnages are long tons of 2240 lb.

[b] Plant shut down seven days (three days Christmas holidays and four Sundays due to production curtailment). Balance of lost time due to shutdown for repairs and accidental causes.

TABLE 4.—*No. 2 General Mill Monthly Composites for December, 1931, Phosphate Recovery Corporation*

CALCULATED FEED

Mesh	Weight, Per Cent	Weight, Cumulative Per Cent	Bone Phosphate of Lime, Per Cent	Insoluble	Recovery, Bone Phosphate of Lime, Per Cent
Total	100.00		21.37		
+ 20	Trace				
+ 28	4.77	4.77	50.32		
+ 35	12.97	17.74	35.78		
+ 48	25.59	43.33	20.36		
+ 65	31.60	74.93	17.12		
+100	18.26	93.19	15.77		
+150	6.08	99.27	11.68		
−150	0.73	100.00	16.44		

ACTUAL MILL CONCENTRATES

Mesh	Weight, Per Cent	Weight, Cumulative Per Cent	Bone Phosphate of Lime, Per Cent	Insoluble	Recovery, Bone Phosphate of Lime, Per Cent
Total	100.00		75.05	7.92	82.4
+20	Trace				
28	6.29	6.29	78.56	3.95	47.9
35	18.54	24.83	79.07	3.58	74.1
48	24.83	49.66	78.37	4.73	87.5
65	28.81	78 47	74.03	8.67	92.6
100	16.23	94 70	70.29	12.81	93.1
150	4.31	99 01	64.63	19.88	91.6
−150	0.99	100 00	47.17	40.01	91.7

ACTUAL MILL TAILING

Mesh	Weight, Per Cent	Weight, Cumulative Per Cent	Bone Phosphate of Lime, Per Cent	Insoluble	Recovery, Bone Phosphate of Lime, Per Cent
Total	100.00		4.93		
+20	Trace				
28	4.31	4.31	38.03		
35	11.26	15.57	13.90		
48	25.83	41.40	3.28		
65	32.45	73.85	1.59		
100	18.87	92.72	1.39		
150	6.62	99.34	1.20		
−150	0.66	100.00	2.66		

SUMMARY OF COSTS

Table 5 gives the details of operating costs, but such costs on an operation of this size are not kept by departments. Power is metered at the plant, but is not metered by departments. Total power used in December, 1931, for milling purposes was 5.344 kw-hr. per short ton of feed handled. The total actual load at the plant averages about 770 hp., including water supply. Total rated motor horsepower is 920, some of the motors being operated intermittently and some at less than capacity.

Table 5.—*Summary of Direct Operating Costs, No. 2 Flotation Concentrator, Phosphate Recovery Corporation, for the Month of December,* 1931

Long Tons Ore Treated, 52,323.38 Long Tons Concentrates Produced, 12,271.00
Short Tons Ore Treated, 58,602.19 Short Tons Concentrates Produced, 13,743.52

	Total for Month	Per Long Ton Feed	Per Short Ton Feed
Operating labor	$1,268.19	$0.0242	$0.0217
Power	3,131.80	0.0599	0.0534
Supplies (operating)	133.18	0.0025	0.0023
Reagents	1,245.03	0.0238	0.0212
Repair labor	329.74	0.0063	0.0056
Repair supplies	614.08	0.0117	0.0105
Sampling	129.01	0.0025	0.0022
Assaying	138.95	0.0027	0.0024
Superintendent, office and local overhead	755.07	0.0144	0.0129
Total	$7,745.05	$0.1480	$0.1322

Segregated into Labor, Supplies and Power

	Total for Month	Per Long Ton Feed	Per Short Ton Feed
Total labor	$1,726.95	$0.0330	$0.0295
Power	3,131.80	0.0599	0.0534
Reagents	1,245.03	0.0238	0.0212
Supplies	747.26	0.0143	0.0128
Superintendent, office and local overhead[a]	894.02	0.0170	0.0153
Total	$7,745.05	$0.1480	$0.1322

[a] Including assaying.

DISCUSSION

(*Samuel Dolbear presiding*)

G. R. M. del Giudice,[*] New York, N. Y.—Does the sequence of reagents make a difference?

C. E. Heinrichs,[†] New York, N. Y.—Yes. It is better to get the alkali in first.

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[*] Lecturer in Mining, School of Mines, Columbia University.

[†] Phosphate Recovery Corporation.

Development of the Grande Ecaille Sulfur Deposit

By Wilson T. Lundy,* New Orleans, La.

(New York Meeting, February, 1934)

The history of the production of sulfur from salt domes in Louisiana and Texas originated with the operations of the Union Sulphur Co. at Sulphur, La., followed by the Freeport Sulphur Co. at Bryanmound, Tex. Including the present development of Grande Ecaille by the Freeport Sulphur Co., which constitutes the tenth dome to be equipped for the production of sulfur in the Gulf Coast, less than 10 per cent of all the known domes have proved to contain sulfur in commercial quantities.

The Grande Ecaille salt dome is in the tidal marsh of the Mississippi River delta in Plaquemines Parish, Louisiana. It is 10 miles southwest of the right descending bank of the river approximately 45 miles below the city of New Orleans and is within 4 miles of the shore line of the Gulf of Mexico.

TERRAIN

Characteristic of the delta region, the terrain consists of a low, flat, uninhabited area of marsh land intersected by many shallow lakes and bayous. With the exception of salt grasses, the region is devoid of vegetation and presents, in all directions, an unobstructed path to the vagaries of the winds. The general character of this area can be best visualized from the mechanism of its formation.[1] Offshore bars formed enclosing long narrow lagoons between the bar and the inner shore line, the lagoons communicating with the sea by means of tidal inlets. The water in the lagoons, being comparatively quiet, favored the deposition of fine debris composed of sand carried into the lagoons by action of the waves, tides and river-borne sediments from the land surface. As this material accumulated the bottoms of the lagoons were built up and eventually salt grasses obtained a foothold. The successions of growth and decay of these grasses formed a brown fibrous mat varying in depth from a few inches to several feet. This mat of vegetation virtually floats on an ooze that is several feet thick and is composed of alluvial sediments and very fine sand, and the whole is underlain by an unctuous clay. The decayed vegetation from the upper layer when dried has a peaty appearance.

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* Vice President and General Manager, Freeport Sulphur Co.

[1] D. A. Johnson: Shore Processes and Shore Development.

EARLY PROSPECTING

The early prospecting of the dome was in search for oil, the discovery of indications of sulfur being incidental to these operations. The mineral rights are jointly held by the Humble Oil & Refining Co., the Gulf Refining Company of Louisiana and the Shell Petroleum Corporation. Prospecting was delegated to the Humble Oil & Refining Co., and initial operations began in July, 1929. While the dome is within 10 miles of the Mississippi River, there existed at that time no practicable means of communication between these points and it was necessary to ship all materials required for the work from Harvey, opposite New Orleans; a distance of about 70 miles. The route followed was out Harvey Canal to Little Barataria Bayou, then through Big Barataria Bayou, Bayou Du Pont, Dupree Cut, Cutler Bayou, Bayou St. Denis and across Barataria Bay into Lake Grande Ecaille, thence to the dome. These bayous and lakes are all shallow, which made the barging and towing of materials not only costly but hazardous.

At two locations in the marsh large wooden mats were used to support the rig and necessary equipment. This did not prove satisfactory because the unstable character of the soil permitted the derrick to settle unevenly, making drilling difficult even for shallow wells. In subsequent operations the derrick was set on piling and the power to operate the equipment was furnished by a diesel electric plant consisting of two 210-hp. units connected to direct-current generators installed on a barge of special design 28 by 90 ft. A barge with pumps and motors mounted on the deck was also provided for the storage and handling of the drilling mud, making the whole equipment, with the exception of the derrick, readily mobile. Under this arrangement a canal was dredged to the location, piling driven and the derrick set in place. The power unit, mud barge and its equipment were then towed into position near the rig and drilling prosecuted.

Insect pests at times added greatly to the discomfort of the employees operating the rigs, and it became necessary to use blowers to enable the men to work. The blowers consisted of Model T Ford motors with airplane propellers attached, each unit guarded by a frame covered with hardware cloth to prevent accidents.

During the drilling operations sulfur water was encountered in the second well at about 1735 ft., and in the third well drilled traces of sulfur were found in the broken cap-rock formation at 1527 ft. Cores from a well drilled at a later date were considered sufficiently rich to warrant the prospecting of the dome for sulfur.

PROSPECTING

At this point, the Freeport Sulphur Co. commenced negotiations with the three oil companies mentioned, which resulted in the acquisition of sulfur rights by the Freeport Sulphur Co. early in the year 1932. A program calling first for prospecting and subsequently for development was immediately inaugurated. Simultaneously, with prospect drilling, a torsion balance survey was made under the supervision of Dr. Donald C. Barton, of Houston, Tex., to determine the size, depth and configuration of the cap-rock area. This information, secured more quickly than possible by drilling alone, was used to advantage in making later prospect well locations, and in selecting sites for power plant and auxiliary buildings. The location of these sites was made off the dome area to insure freedom from the surface subsidence, which is expected to result from the extraction of sulfur by the Frasch process. The prospecting of the dome for sulfur began in April, 1932, and within a year 18 wells were drilled and sampled.

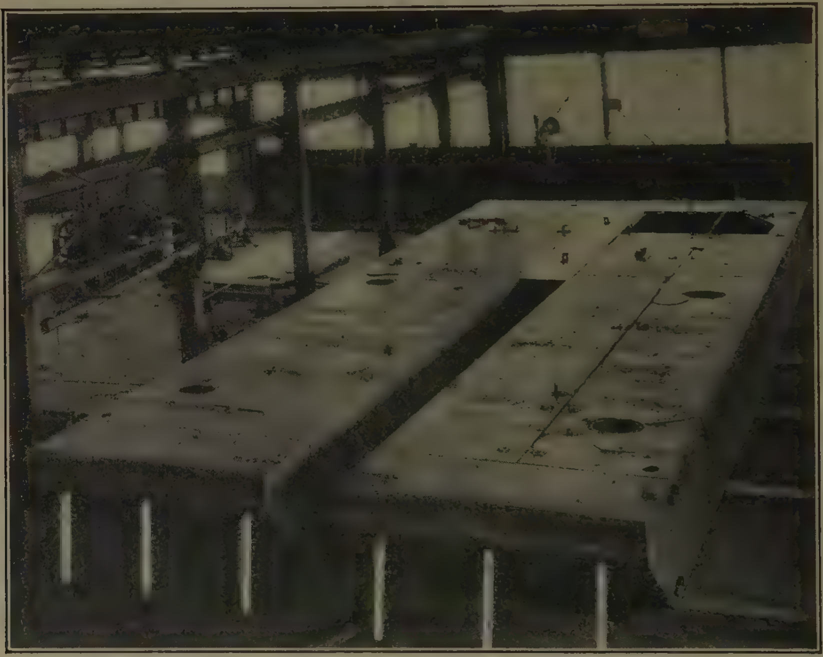

FIG. 1.—DRILLING BARGE DURING CONSTRUCTION.

Floating Drilling Equipment

To overcome the drilling difficulties encountered by the Humble Oil Co., the use of floating drilling equipment was employed. Two welded steel barges were specially designed to support and house the entire drilling units. Each unit consisted of a 96-ft. derrick, 100-hp. engine for the drawworks, 50-hp. engine for the slush pump, electric lighting system, mosquito blowers, water pump, tools and equipment for rotary drilling rig. The derrick and draw-works engine were mounted on an 8-ft. steel substructure, which allowed space beneath for the location of the pumps and other equipment. Five gasoline engines were required to furnish power for each unit. These drilling barges, measuring 36 by 80 by 6½ ft., were divided into watertight compartments by longitudinal and transverse bulkheads, which also served to strengthen them sufficiently to resist the shocks and jars incident to rotary drilling. The two rear compartments served as slush pits to hold the drilling mud, the forward compartments serving as water-ballast tanks to maintain the barge on an even keel. An opening 4 ft. wide, extending from the bow to the center of the barge, provided a means for moving on or off a well location where casing had been set. Four 36-ft. spuds made of heavy steel pipe, one at each corner of the barge, were used to hold it in position over a location. In operating the drilling barge the procedure used was to dredge a canal 45 ft. wide by 6 ft. deep to the location. The canal was then enlarged at this point to 70 ft. in width for a distance of 175 ft.,

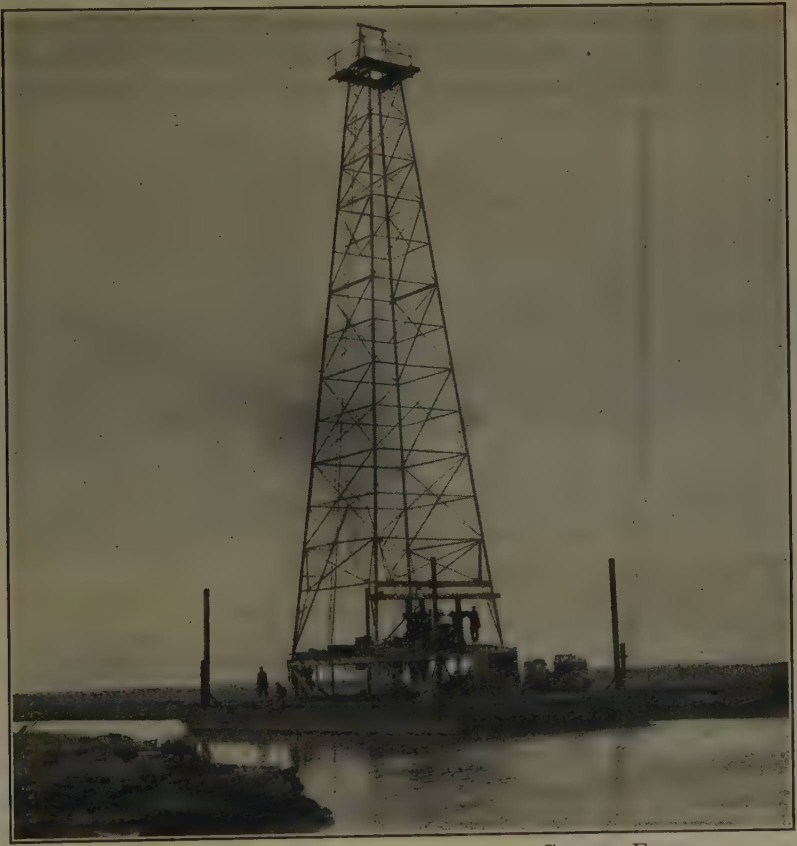

Fig. 2.—Completed barge on location at Grande Ecaille.

in order to facilitate the handling of material barges near the drilling location. Upon completion of the canal the drilling barge was towed into position, anchored in place by means of the spuds, and drilling begun immediately. This method of operation was found to be particularly well adapted to marshy and shallow lake locations at Grande Ecaille, with a great saving in expense and time of moving and setting up.

The sedimentary formations overlying the cap rock consist of gumbo, sand, sandy shale and a few boulders, and offered little resistance to drilling. At each location, approximately 80 ft. of 15½-in. surface casing was set, then a 13¾-in. hole was drilled to the cap rock, where 10-in. casing was set and cemented at a depth usually of about 1250 ft.

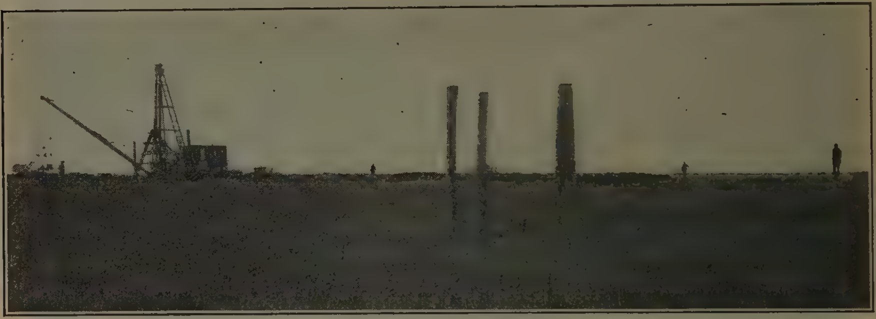

FIG. 3.—PLANT SITE, GRANDE ECAILLE.

The thickness of the cap rock in the wells drilled averaged about 250 ft., consisting of limestone, calcite, gypsum, anhydrite and sulfur, and with traces of pyrite, barite and celestite.

Sampling

In the prospect drilling, the entire thickness of the cap rock overlying the salt plug was either cored or sampled by other means in order that a maximum amount of the formation might be recovered for observation and analysis. Two sampling methods were found to be practical at Grande Ecaille, the reverse return or air-lift system and the continuous operation of core bits used in conjunction with a sand pump. For coring, a specially designed 5-ft. core barrel cutting a 9⅞-in. hole and taking a core 6 in. in diameter was used. After each run of the core bit the sand pump recovered the part of the formation that was drilled but not picked up by the core barrel. The sand pump was also found to be essential not only in aiding the recovery of the formation drilled, but also in serving to keep the hole clean, because the drilling fluid would not carry the cuttings to the surface. The two samples from each 5-ft. interval were then dried, weighed and assayed for sulfur content. In the final calculations for sulfur content the two samples were combined according to their respective weights, thus giving a fairly accurate figure for percentage of sulfur in each interval. Seven wells were sampled in this manner with an average recovery of 56 per cent of the formation drilled.

In sampling by the method of reverse returns, the drilling fluid, consisting of clear water and the cuttings, is lifted through the drill pipe by means of an air-lift. A specially designed swivel is used to provide means for holding the ½-in. air line in the drill stem and also to provide a hose connection for conveying the cuttings to the sampling equipment. A separate barge was used to house the sampling equipment. A splash box with an air vent on top received the fluid and cuttings and permitted the escape of the air, and an opening in the bottom allowed the sample to pass immediately to the sampling equipment. A trough with a 16-mesh screen below the splash box caught all large cuttings. The flow passing through this screen was conducted to a flume where, by means of slotted openings, it was exactly halved three times, thus leaving one-eighth of the sample to be retained in the settling box. The settlings and cuttings were then dried, weighed and assayed in the same manner as in the coring method. The drilling tool was either a fishtail bit or a ball-bearing rock bit with the water courses enlarged to permit the passage of fluid and cuttings without excess friction. Air at 150 lb. pressure was used, with a flow of from 80 to 100 gal. of water per minute. By this means of sampling the recovery of the formation approximated 100 per cent and the speed attained was considerably greater than where core bits were used.

FIG. 4.—CONSTRUCTION CAMP.

RESERVES

The cap-rock area within the 2000-ft. contour as outlined by the torsion balance totals 1100 acres. Of this area, only 209 acres have been drilled and proved to contain sulfur in commercial quantities. The remaining undrilled area is considered a promising potential but has not been proven.

The Plant

During the prospecting period the engineering department of the Freeport Sulphur Co. prepared and carefully studied several tentative plans for equipping this property, and in this work the J. F. Coleman Engineering Co., of New Orleans, and the J. G. White Engineering Corporation, of New York, assisted and were retained as consultants both for the preliminary design and during the construction period. The chief problem in constructing and equipping a property with terrain conditions as described above lay in the selection and design of foundations. The plan finally adopted, after exhaustive tests, consisted of heavy reinforced concrete mats supported by piling for the foundations for buildings, and hydraulically made fills for the vat foundations and mining area. The vats are also supported by piling.

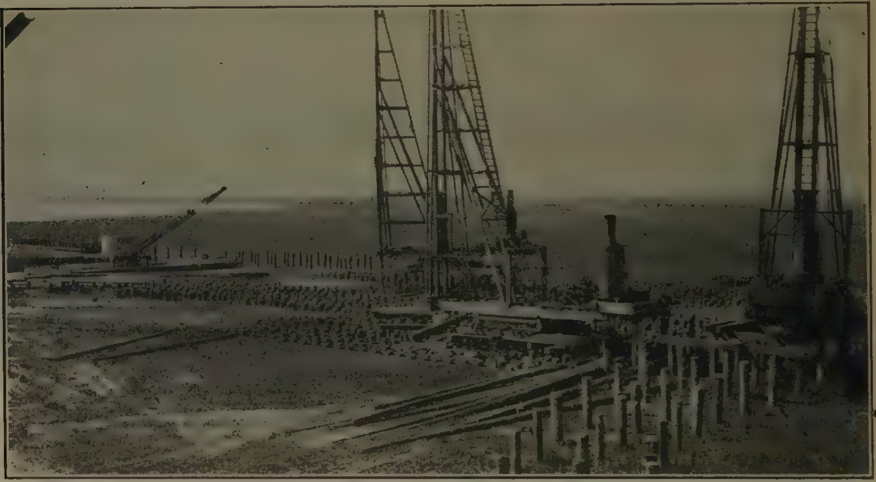

Fig. 5.—Driving piles for plant foundation. Travel piles in foreground were used later as supports for runways for transportation of materials.

Grandeporte.—A site on the Mississippi River accessible by both rail and highway was selected and purchased as a base for receiving and handling materials and for the general transportation of sulfur. The river terminal has been named Grandeporte, La., where the mine office, laboratory and a model industrial village are being constructed, with a school, community center, parks and recreation facilities for the employees and their families. To provide for rapid and economical transportation from Grande Ecaille operations, a canal approximately 100 ft. wide, 9 ft. deep, and 10 miles long was dredged, involving the movement of some 2,000,000 cu. yd. of dirt. Grandeporte is reached by the New Orleans and Lower Coast Railroad, a branch of the Missouri Pacific Railroad

System. A river dock 1000 ft. long has been provided for the accommodation of vessels having a draft of as much as 35 feet.

Foundation Conditions.—The geological formations in the alluvial valley of the Mississippi River differ widely from those of the adjacent territory. The valley was originally an estuary extending to Cairo, Ill. As it became filled by the deposits of silt, the river carved out its channel in a meandering course. Below Baton Rouge, La., the river is in the center of the alluvial valley. Foundation conditions in this valley are difficult, vary considerably from place to place, and usually improve as the river is approached. Fairly satisfactory soil conditions, where pressures up to 1200 lb. per square foot can be used, are encountered at various places, while at others this loading would produce prohibitive settlement. Structures of any magnitude must therefore be supported on piling. In the lower valley, a short distance from the river, this type of foundation is essential because the soil has no supporting power. The various formulas for loading were not employed therefore and foundation designs were based upon information gained from driving and loading tests. Numerous such tests were conducted in this section to determine the feasibility of using piling foundations, and also to select the best sites for plant and other buildings. It was found that no stratum existed sufficiently close to the surface to be of any value for supporting the piling and that the friction between the soil and the piles must be

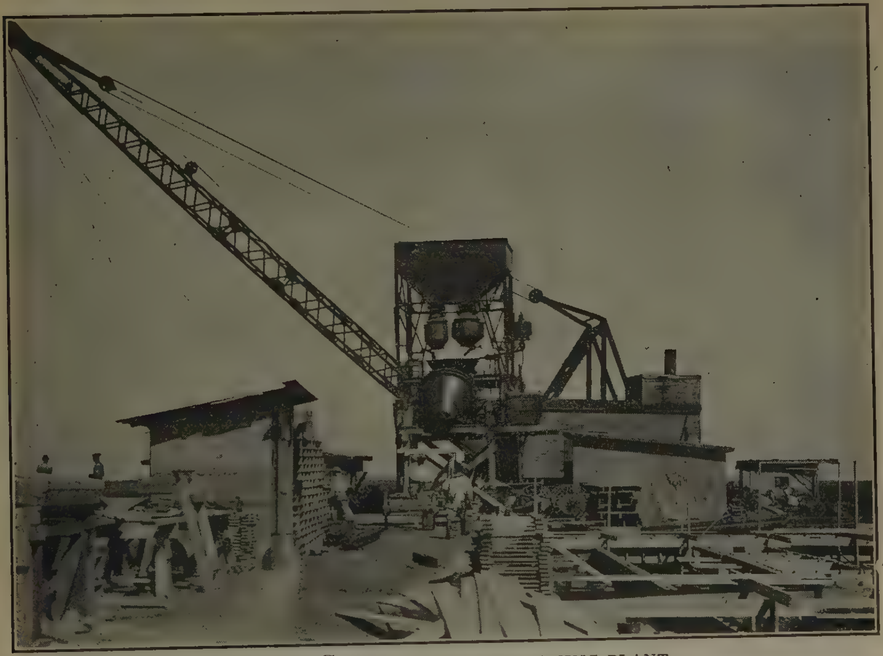

FIG. 6.—CENTRAL CONCRETE-MIXING PLANT.

depended upon solely. To meet this condition sufficient penetration was required to insure a frictional resistance capable of safely sustaining the imposed conditions of loading without settlement.

Piling.—Piling tests were conducted to determine definitely the supporting power of piling in this type of soil. From the behavior of the piles during the driving and under these load tests, 75 ft. was decided upon as the proper length for use in the main structures with a safe load limit of 8 tons per pile. Since the supporting power of a group is considerably less per pile than an equal number widely scattered, the load limit as adopted was tested more accurately and it was found that no settlement occurred when a weight of 10 tons per pile was placed on a group of 4 in a cluster of 16.

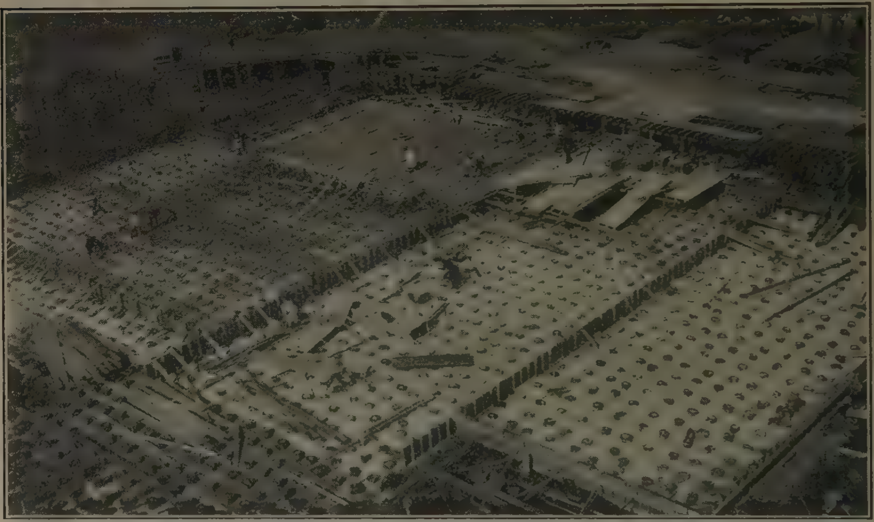

Fig. 7.—Plant foundation piling. Mat being poured.

During the driving of the piling it was found that the upper 45 ft. of the soil offered practically no resistance because the piles under their own weight and the weight of the hammer penetrated almost uniformly 45 ft. before a blow was struck. In spite of the soft character of this material, it has a very noticeable tendency to set up within a very few hours. Where driving was continuous a penetration of 8 in. per blow for the first 20 ft. was obtained, whereas only a ½-in. penetration was recorded for piles left standing overnight with the hammer lowered into position ready for use. The hammer was a Vulcan No. 1 weighing 10,000 lb., with falling parts weighing 5000 pounds.

The concrete mats, while designed to distribute the load uniformly, were given adequate strength to provide for any variations in the individual supporting power of the piles. Future requirements were also taken into consideration, since it is impossible to satisfactorily alter these foundations when once they are installed. The foundation of the boiler plant required some 3500 untreated piles, 75 ft. long, driven on approximately 2 ft. 8 in. centers. The total requirements for piling for the project will run in the neighborhood of 18,000, varying in length from 40 to 80 ft. and approximately 10,000 cu. yd. of concrete was used to provide adequate foundations for the numerous buildings.

Water.—Since the mining of sulfur involves the heating and handling of large quantities of water, a major consideration in the design of a plant is the selection of a dependable source of water and fuel. As all lake and bayou water in the area surrounding the plant is high in salt content and encrusting solids, and such well water as can be obtained is brackish, it was imperative that some other source be developed. Investigation developed that water comparatively low in salt content could be obtained from the Mississippi River and accordingly an earthen reservoir to hold fifty million gallons was constructed for settling the turbid water from this stream before pumping it to the plant. An intake pump station for delivering the river water to the reservoir and a similar pumping plant at the reservoir for delivering the water to the power plant were required. A pipe line approximately 9 miles long conveys the water to the plant and the power required for pumping is transmitted over an

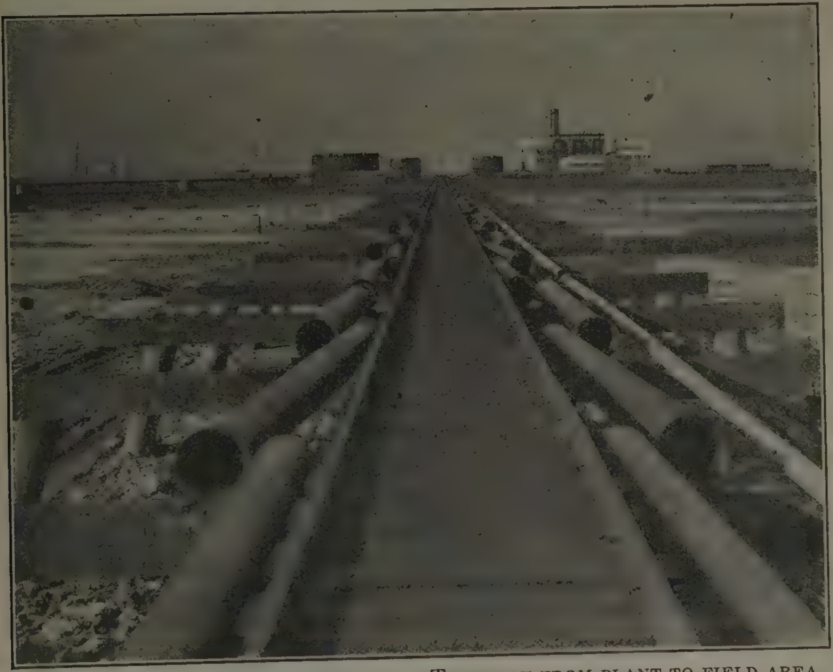

FIG. 8.—PLANT AUXILIARY BUILDING. TRAMWAY FROM PLANT TO FIELD AREA.

electric transmission line of the same length. Storage for two million gallons of water is provided at the plant to maintain continuous operation should it be necessary to make repairs to the reservoir, pumping equipment, or pipe line. The Mississippi River water, while very low in salt, contains sufficient encrusting solids to make treatment advisable in order to prevent objectionable deposits in boilers, pipe lines and heaters. The method adopted includes hot lime-soda treatment for the boiler water and hot lime treatment for the water used in the mining operations. Soda ash is not required for the mine water, as there is no concentration of the water such as occurs in boilers, and the permanent hardness, remaining in solution, does not cause deposits in the mine system.

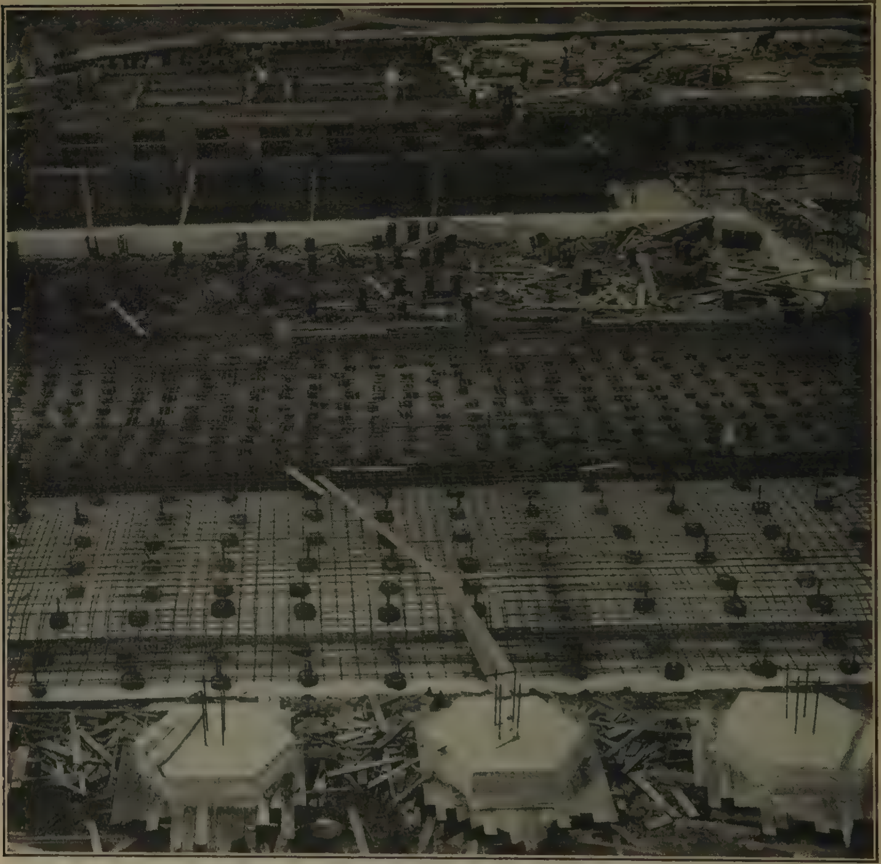

FIG. 9.—REINFORCING IN CONCRETE MATS.

Fuel.—Fuel oil is received by tanker and is discharged into storage consisting of three 55,000-bbl. tanks, located 1500 ft. from the river and built on an earth fill topped with sand. The fuel oil will be conveyed to the plant by barges operating through the dredged canal and discharged into two 15,000-bbl. firing tanks. The firing tanks as well as the water-storage tanks were built on piling foundations capped with concrete mats. The fuel oil flows from the firing tanks to the fuel-oil pumps in the plant by gravity and is heated and pumped to the burners under high pressure.

Plant Construction.—The power plant and all other permanent buildings are constructed of steel with corrugated asbestos roofing and siding, using aluminum bolts and clips of high tensile strength for fastenings, Owing to the exposed location, the steel frames are all designed for 125-mile winds. All window sash is of galvanized steel and gutters and downspouts are aluminum, to minimize the effects of corrosion. Material from the canal to the plant and other buildings was transported on ramps built on piling and decked with 3-in. planks, the supports being first used as travel piling for the drivers during construction of the foundations. A tractor and trailer were used on the ramps to handle the material to the point of erection and trucks hauled the concrete from the mixing plant to the various buildings. The floor of the plant is at an elevation of 12 ft. above mean Gulf tide, or 11 ft. above the surrounding marsh, this being considered ample to protect the plant during storms and high tides. The piling was cut off about 1 ft. above the marsh, capped with a thick concrete mat, and piers constructed to support the plant floor and equipment. The space between the floor and the mat furnishes a storage for water and provides a convenient suction inlet to the cold-water pumps supplying the water-treating plant.

Fig. 10.—Construction of transmission line.

Considering the unfavorable location, construction costs have been kept relatively low. The foundations for the auxiliary buildings, such as the machine shop, blacksmith shop and warehouse, averaged $2.11 per square foot of floor area. The plant building, which was constructed on a concrete mat 2 ft. 6 in. thick, with extra heavy piers closely spaced supporting the equipment and plant floor, equivalent in area to three

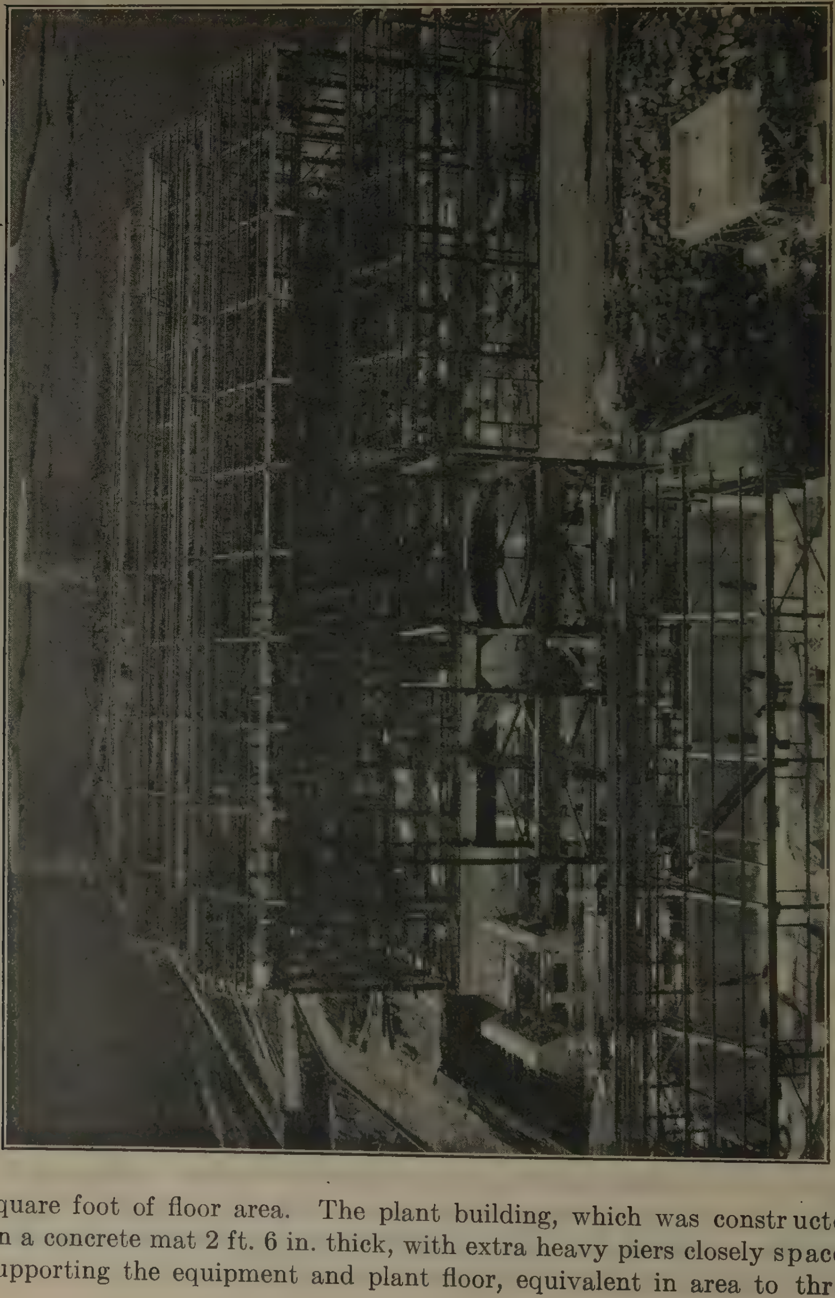

FIG. 11.—STEEL WORK ON PLANT

supporting floors, ran $6.60 or an equivalent of $2.20 per each square foot of floor area. Superstructures fabricated and erected in place complete, including siding, roofing, windows and doors, averaged 5 cents per cubic foot of enclosed volume. Concrete costs were $17.30 per cubic yard including form work and reinforcing.

Plant Equipment.—In passing through the water-treating plant the water is heated, treated, aerated and filtered. The equipment consists of three treating units each having an exhaust-steam jet heater and vent condensor, sedimentation tank and two filters. Exhaust steam from the various prime movers in the plant is utilized in preheating the water for treatment. Chemicals are automatically proportioned to the amount of water flowing through the treating plant. The water is heated to about 218° F. at the water-treating plant, which is adjacent to the boiler plant.

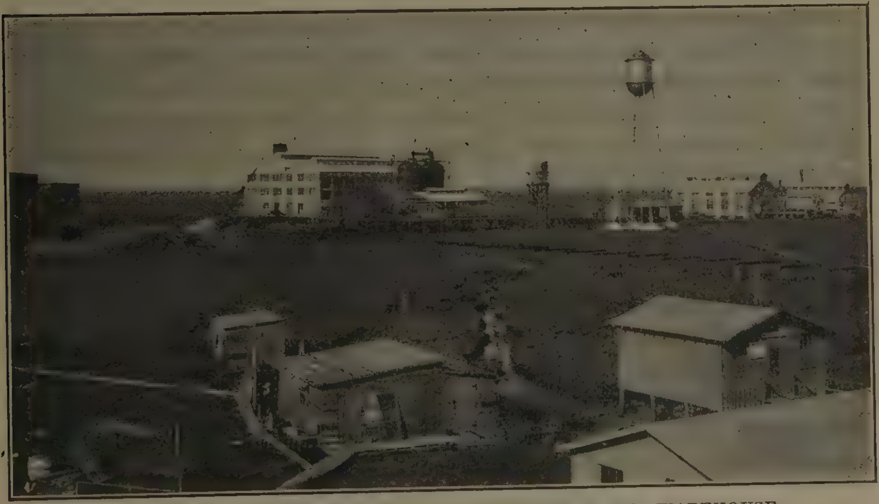

FIG. 12.—PLANT, MACHINE SHOP, BLACKSMITH SHOP AND WAREHOUSE.

High-pressure heater supply pumps take the water from the treating plant and pass it in direct contact with live steam at 120 lb. pressure in high-pressure heaters, where the temperature of the resulting mixture is raised to approximately 350° F. and pumped to the field for mining purposes. All pumping equipment is of turbine-driven centrifugal type, turbines being considered most reliable in maintaining the continuity of operation that is essential to sulfur mining.

Six 860-hp. bent water-tube Stirling boilers with air-cooled furnaces are used in the boiler room for generating steam and are designed to operate at 200 per cent of builder's rating. The boilers are set high to provide ample furnace volume for combustion of oil, gas or pulverized solid fuel and to provide for rear firing. Space and foundations have also been provided for equipment and facilities for pulverized fuel. Mechanical burners are used and automatic valve-in-head soot blowers were installed in each boiler. Draft is provided by three large induced-draft fans driven by turbine gear and a short steel stack is provided to conduct the flue gases to the atmosphere.

Three 750-kw. noncondensing turbogenerators provide current for drilling wells, pumping water, loading sulfur and all other operations incident to the mining of sulfur. Three 800-lb. high pressure air compressors supply air to the field for pumping the sulfur wells. The cost of steam used in driving the generators, compressors and pumps is negligible because all exhaust steam is returned to the mine-water system through the low-pressure heaters and water-treating plants.

In order to maintain a relatively low formation pressure, all the water injected into the formations must be bled from the mine. This waste formation water is conveyed to a treating unit approximately 1000 ft. from the power plant, where the objectionable sulfide impurities are removed before disposal.

Hydraulic Fills.—The low marshland constituting the mining area has been filled to an elevation of from 4 to 8 ft., and this work will be carried on progressively by a 12-in. diesel electric dredge. Hydraulic fills also include the area surrounding the plant, in order to preserve the piling as well as an operating convenience. The area selected for sulfur storage has been filled to a height of 12 ft. to protect stocks from water. A tramway for the transportation of materials has been constructed on piling bents to connect the mining area and the plant area, a distance of approximately 4000 ft. These bents also carry the various pipe lines required for steam, water and air. The completed project will involve the dredging of more than two million cubic yards of earth fill, and it has been found necessary to excavate to a depth of 40 to 50 ft. in order to obtain suitable material.

With the completion of the field fill, the use of the drilling barges and floating equipment has been discontinued and the derricks have been erected on mats. Motor drives have been substituted for the gasoline engines, power being supplied by a high line from the plant.

Mining, Transportation and Loading

In the mining of sulfur, fuel constitutes one of the major items of cost. High plant efficiency is, therefore, of great importance. It is estimated that an 80 per cent boiler efficiency and 77 per cent over-all efficiency will be obtained in the plant. Of the total heat recovered from the fuel, approximately 3 per cent is usually required for the handling of liquid sulfur in pipe lines and in relay stations on the surface, less than 1 per cent for auxiliary power to drive generators, pumps, compressors, etc., leaving approximately 97 per cent for the injection into the formation. Operating the boilers at 200 per cent of the builder's rating, approximately two million eight hundred thousand gallons of water per day having a temperature of 350° F. can be furnished to the mining area through the booster pumps at a pressure of 250 lb. per square inch. This quantity of water is made up approximately of 75 per cent mine water and 25 per cent boiler water.

The productive capacity of any Gulf Coastal sulfur mine is highly variable. With similar plant capacities, the production of one mine rarely equals that of another. The production varies from day to day and no area or even individual wells yield identical results. This variation of production is primarily due to the ever-changing thermal efficiency of the rock structure into which the water or heat is injected. Many factors upon which little or no control can be exercised enter into the thermal efficiency of individual deposits, areas or wells. The major factors are probably the physical nature of the rock structure, rapidity of subsidence, reserves, concentrations and recovery of sulfur. An estimate of the rate of production from Grande Ecaille is, therefore, difficult to determine.

The sulfur produced at the individual wells is lifted to the surface by the means of an air-lift. The molten sulfur is then conveyed from the wells to the relay station, whence it is pumped to a vat and allowed to cool and solidify. The weight per square foot of area of a sulfur vat 40 ft. high averages approximately 2¼ tons, and to prevent excessive settlement the supporting power of the vat area has been increased by 60-ft. piling placed on 8-ft. centers.

The solidified sulfur in the vats, when broken down by blasting and loaded on a belt conveyor by means of electric shovels, is discharged into barges for transportation. The broken sulfur is barged through the canal to the river. At this point, the sulfur is unloaded from the barges by means of an electric crane and can either be placed in storage or conveyed by a belt over the docks into the holds of vessels or to barges in the Mississippi River at the rate of 500 tons per hour.

Fluorspar Deposits in Western United States*

By Ernest F. Burchard,† Washington, D. C.

(New York Meeting, February, 1933)

Fluorspar is found in most of the states from the Rocky Mountains westward, and commercial production of the mineral has been reported from Arizona, Colorado, Nevada, New Mexico, Utah and Washington. The map in Fig. 1 indicates the general distribution of many of these deposits but it is by no means complete.

In the summer of 1926 the writer examined certain deposits of fluorspar in Arizona, California, Colorado, New Mexico and Washington; in November, 1927, revisited the Jamestown, Colo., district, and in earlier years examined deposits in Colorado and New Mexico. Based on notes of his own and on descriptions and other data by H. A. Aurand, H. W. Davis, R. B. Ladoo, V. C. Heikes, W. D. Johnston, Jr., and by producers of fluorspar, he prepared the following brief descriptions of deposits and estimates of reserves of fluorspar, originally for the use of the subcommittee on fluorspar of the Joint Committee on Foreign and Domestic Mining Policy and on Industrial Preparedness respectively of the Mining and Metallurgical Society of America and the American Institute of Mining and Metallurgical Engineers.

Descriptions of many fluorspar deposits in various western states and districts have been published. It is not feasible to summarize this literature here, but the appended bibliography will indicate to the interested reader what papers are available.

ARIZONA

Fluorspar has been reported as occurring in Arizona in the following localities: near Duncan, Greenlee County; near Naco, Tombstone, and Paradise, Cochise County; Black Mountain, Mojave County; 25 miles southwest of Tucson, and at Helvetia and Silver Bell, Pima County; near Mammoth mine, Pinal County; in Yavapai County, 7 miles from Aguila, and in the Bradshaw Mountains; and near Dome, Yuma County. Production has come principally from the Castle Dome district, Yuma County, with minor quantities from Greenlee, Pima and Yavapai counties. The writer examined deposits in the Castle Dome district, Yuma

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† Geologist, United States Geological Survey.

County, and near Duncan, Greenlee County; of these only those near Dome appear to be of commercial importance.

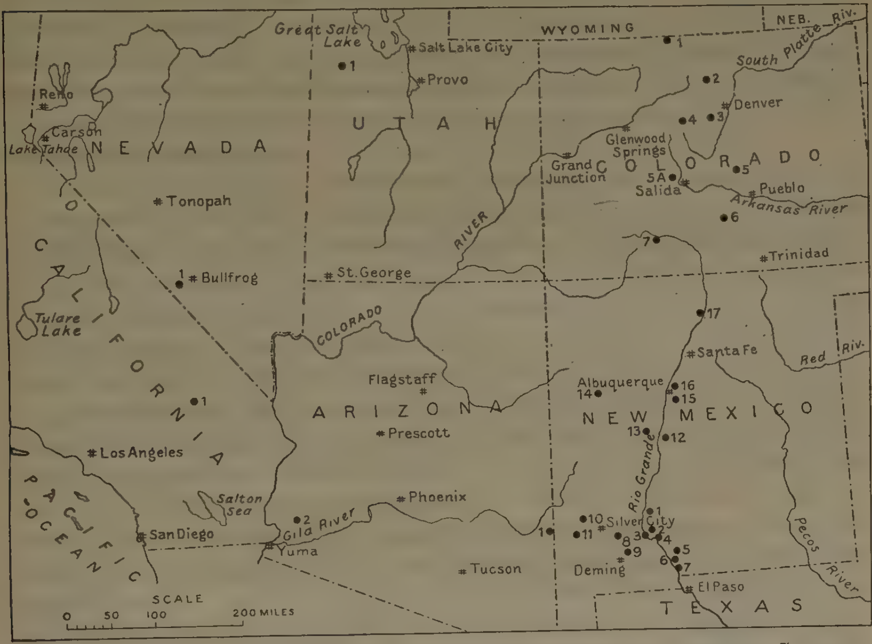

FIG. 1.—LOCATIONS OF IMPORTANT FLUORSPAR DEPOSITS OF WESTERN STATES.

ARIZONA

1. Greenlee County (8 to 16 mi. NE. of Duncan)
2. Yuma County (25 mi. NNE. of Dome)

CALIFORNIA

1. San Bernardino County (5 mi. SE. of Afton)

COLORADO

1. Jackson County (3½ mi. NE. of Northgate)
2. Boulder County (Jamestown)
3. Jefferson County (Evergreen)
4. Park County (Jefferson)
5. El Paso County (Duffields)
5A. Chaffee County (Mt. Antero)
6. Custer County (16 mi. SE. of Westcliffe)
7. Mineral County (Wagon Wheel Gap)

NEW MEXICO

1. Sierra County (5 mi. SE. of Hot Springs)
2. Dona Ana County (20 mi. N. of Hatch)
3. Sierra County (20 mi. NW. of Hatch)
4. Dona Ana County (1 mi. E. of Heathden)
5. Dona Ana County (4½ mi. S. of Organ)
6. Dona Ana County (5 mi. NE. of Mesilla Park)
7. Dona Ana County (10 mi. E. of Mesquite)
8. Luna County (17 mi. NE. of Spaulding)
9. Luna County (6 mi. NW. of Mirage)
10. Grant County (30 mi. NW. of Tyrone)
11. Grant County (33 mi. N. of Lordsburg)
12. Joyita Hills
13. Sierra Ladrones
14. Zuni Mts.
15. Manzano Mts.
16. Sandia Mts.
17. La Madera

NEVADA

1. Nye County (6 mi. SE. of Beatty)

UTAH

1. Tooele County (20 mi. SW. of Clive)

Deposits near Dome, Yuma County

The Castle Dome district is about 25 miles northeast of the village of Dome, on the Southern Pacific Railroad. The mining district is on a terrace and foothill spurs on the west side of the Castle Dome Range at an altitude of 1400 ft. according to a U. S. Geological Survey bench mark. The Castle Dome mountains are composed of flows of light colored volcanic rocks that intrude and overlie gneisses and slates of probable pre-Cambrian age.

The mineralized veins are mainly in the volcanics and seem not to be along definite contacts. The rocks are much jointed and fractured and so easily broken with pick and bar that shaft sinking and tunnel driving is done with the consumption of little explosive. Fluorspar is the principal vein material accompanying galena, for which the mines are worked. Mining has reached depths of 600 ft., one mine having produced, according to local report, silver-lead ore valued at more than $2,000,000.

In October, 1926, the mine operated by A. R. Haack afforded a typical example of the deposits in the district. This mine was open to a depth of more than 300 ft. on one of the veins. The vein pitches steeply toward the northeast and reaches a thickness of 3 to 4 ft. Levels driven on the vein show ore to distances of more than 300 ft. from the shaft on the 200-ft. level and the vein seemed to grow richer with increasing depth. The vein is not a single structure but branches, and in places contains wedges of country rock inclosed by mineralized material. The succession of minerals from the wall rock to the interior of the vein is fluorite and galena, fluorite, barite and selenite, ankerite, and calcite with more or less silica all the way through. In places fluorite constitutes 40 to 60 per cent of the vein material. The galena is intimately mixed with the fluorite and cannot be completely and easily separated from it by cobbing, although much good lump fluorspar may thus be obtained. The galena was reported to yield about 31 oz. of silver to the ton. The mines are dry to depths of 200 to 300 ft. and below this depth there is not sufficient water for concentrating the ore, consequently a method of air separation has to be employed. The apparatus used is a small table having perforations through which air is forced from below. From the ore treated on this table a fairly good recovery of galena is obtained, but fluorspar in the form of fine to coarse sand is mixed with silica and all other waste, and is not in good condition for recovery. Good lump spar is available and a small tonnage was awaiting shipment at the time of visit. A few lumps were so clear as to suggest the possibility of yielding optical fluorite. If subjected to standard milling practice, gravel spar of steel-making grade could be produced. The handicaps to utilization of this abundant byproduct are lack of water and distance from railroad and from large markets.

Fluorspar is found in several other mines and prospects in the Castle Dome district, among them properties of Mrs. Eliza DeLuce, A. G. Hull and Althec Modesti. Some fluorspar was shipped from here prior to and during the World War. Some that was shipped was rejected because of insufficient cleaning. Autotruck haulage to the Southern Pacific Railroad at Dome is about 25 miles over a fair mesa road practically all downgrade. In 1926 the cost of haulage was about $6 per ton, in addition to which there was a ferry charge of $2.50 when there was water in Gila River.

The total recorded production of fluorspar in Arizona to date is 1742 tons, and the reserves of available spar are estimated at approximately 90,000 tons, of which approximately 30,000 tons are of high-grade lump spar and 60,000 tons are of gravel spar suitable for metallurgical flux. The production of gravel spar is, however, provisional upon hauling crude spar to Gila River, where a washing plant might be built. The objections to this would be the cost of hauling probably three times as much waste as recoverable spar.

California

Fluorspar is reported to occur in seven or eight places in California, most of them widely scattered within the southern half of the state. Most of these occurrences are in connection with other mineral deposits and not of commercial value. Certain of these have been noted by D. F. Hewett in the New York and Clarke Mountains in San Bernardino County. Deposits in small veins are reported by the State Division of Mines(16)* as occurring south of Baxter, 4 miles east of Nipton, and 25 miles south of Cima.

Afton District

Efforts to show up a quantity of fluorspar have been made in the desert 2½ to 5 miles south-southeast of Afton, San Bernardino County, and this district was visited by the writer in October, 1926. Several groups of claims have been located there in an area of low hills and ridges south of Mojave River. The country is well dissected by canyons that can be traversed by autotrucks, although with difficulty in places where the sand is deep and dry. The relief is less than 500 ft. in the vicinity of the claims and the altitude is about 2200 ft. In all, there are said to be about 40 lode claims of approximately 20 acres each, within an area nearly 3 miles square.

The fluorspar is associated with fine-grained, grayish where fresh, andesitic rocks, finely porphyritic in places. The andesitic rocks occur as intrusions and as flows over "red beds," a coarse sandstone, sometimes conglomeratic. In places a white, hard, volcanic ash overlies the red beds and is in turn overlain by andesite. There are also red granite and basalt in the region, and limestone is reported.

The volcanic rocks are closely fractured, forming breccias in places, and the brecciated material along certain zones contains much crystalline fluorite, which not only forms veins and cementing material in the crevices but has replaced the rock to some extent. The veins are from a few inches to 4 ft. thick and the fluorite-bearing zones of brecciated rock are of irregular size and shape, ranging from 1 to 50 ft. thick. Some of them have been traced for a distance of 3000 ft. Erosion shows the depth of the deposits sometimes as much as 50 ft. In the richer parts of the fluorite-bearing zones there may be locally 10 to 40 per cent of fluorspar in the rock, but such areas are not extensive. Much silica and calcite are present but metallic sulfides were not noticed. The silica occurs both as quartz and as white, opaque fluoritic material. A little prospecting has been done by means of cuts and short tunnels. There is a little good crystalline spar of greenish and purplish colors, but hardly enough to warrant commercial production of lump spar.

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* Superior figures in parentheses refer to the bibliography at the end of the paper.

A report on the properties by Smith, Emery & Co., based on an examination made in April, 1932, includes 17 analyses, which showed a range in calcium fluoride (CaF_2) of from 35 to 86 per cent; silica (SiO_2) from 8 to 44 per cent, and calcium carbonate ($CaCO_3$) from 2 to 39 per cent. A report by E. C. Clark and S. Meacham in May, 1920, showed some ore carrying as high as 95.6 to 97.7 per cent of calcium fluoride, but it is stated that ore of this grade is small in quantity. A report by the State Division of Mines(16) shows 90.78 per cent calcium fluoride, 5.40 per cent silica and 2.26 per cent aluminum oxide.

The Afton ore is of an unusual type and no experience in mining or milling it can be cited. It is questionable whether it would pay to carry such a quantity of ore and waste to water, and if not a mill would have to be built near the deposits and water obtained either from a local well, if possible, or from a well in the channel of Mojave River and pumped 2½ miles or more against a head of about 700 ft. A road 3 miles or more long would have to be ballasted to the Union Pacific Railroad, crossing the bed of Mojave River, which is dry on the surface most of the time but is subject to sudden floods in winter, and is said to have an underflow at all times. The report by Smith, Emery & Co. indicates that much of the fluorspar ore is of milling grade and that important tonnages are likely to be proved by further prospecting.

Data on which to base estimates of tonnage of fluorspar in the area south of Afton are meager, but, based on a depth of 50 ft. for all the veins and a fluorspar content of 15 per cent for most of the veins and of 10 per cent for the largest brecciated deposits, there is derived a total of about 75,000 tons. No fluorspar has been recorded officially as having been produced in California, but people interested in the claims near Afton say that a carload of picked spar was shipped from there several years ago.

COLORADO

Fluorspar is known to occur in many places in Colorado as a gangue with metalliferous ores and it has been mined on its own account at ten or more places. Of these commercial occurrences, deposits in three places are of outstanding importance; *viz.*, those at Wagon Wheel Gap, Mineral County; Jamestown, Boulder County; and North Gate, Jackson County. Other places at which the mineral occurs in smaller, but probably commercial quantities, are Duffields and Cather Springs, El Paso County; Barstow, Ouray County; Antelope Creek, Custer County; Jefferson, Park County; Evergreen, Jefferson County; and Brown Canyon, Chaffee County, and there are probably other places at which it might be mined if transportation facilities were available.

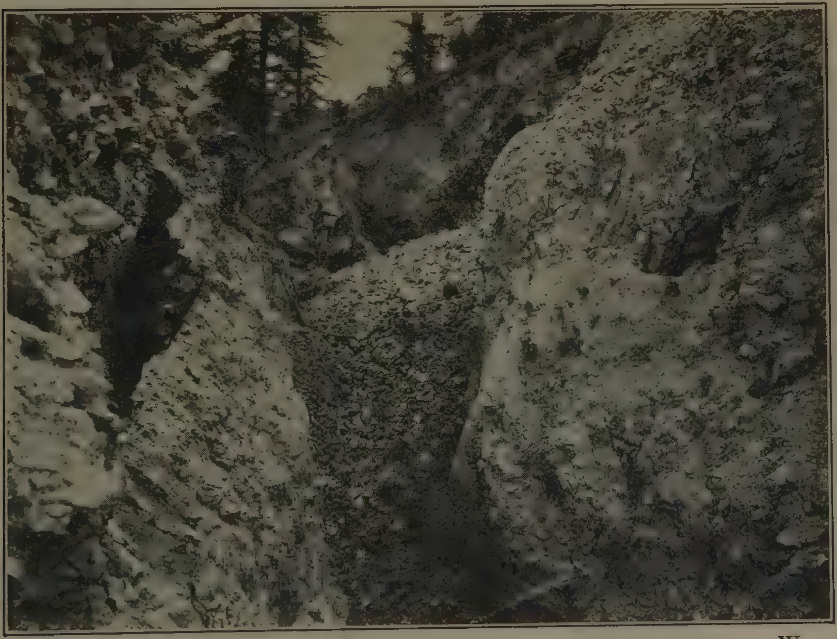

FIG. 2.—OPEN CUT ON OUTCROP OF NEARLY VERTICAL FLUORSPAR VEIN AT WAGON WHEEL GAP MINE.

The vein occupied the space between the walls of cut which show pinches and swells in the width of the vein. On the right-hand wall is a well marked solution (?) breccia containing kaolinite and decomposed country rock.

Wagon Wheel Gap

A large vein of fluorspar cuts rhyolitic tuffs and breccias of Miocene age on the east side of Goose Creek about 1¼ miles above the junction of this creek with the Rio Grande at Wagon Wheel Gap.

The deposit follows a zone of sheeted rhyolite containing one to several veins of fluorspar more or less parallel to the walls and separated by slices of country rock (Fig. 2). The vein zone is nearly vertical; the hade is generally steep toward the south but near the upper surface it is steep toward the north. The wall rock is so much altered that its original character is hardly distinguishable. The associated minerals are pyrite barite, quartz, calcite, kaolinite (?), clay, and a hard, white materia consisting of silica and fluorite, which grades into and incrusts masses of fluorite. This siliceous material is finely granular in places and in others is smooth and amorphous like chalcedony, and it is so near the specific gravity of fluorspar that it is not easily removed in milling. The fluorite and pyrite seem to have been deposited at nearly the same period. The deposition of the vein minerals seems to have occurred locally in open spaces, for there is much mammillary fluorite and well crystallized barite. The colors of the Wagon Wheel Gap fluorspar are generally white or pale green and faint purple. Radiating, fibrous structure with concentric banding is common. Pockets of a smooth, white kaolinite (?) are numerous near the surface and cause the surrounding ore to be loose and subject to slipping down unexpectedly in mining.

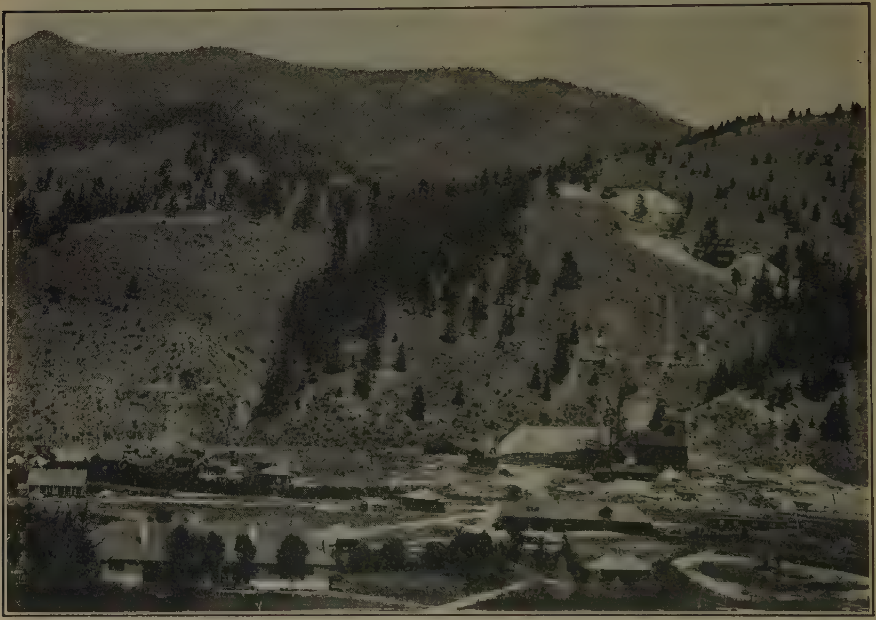

FIG. 3.—FLUORSPAR MINE OPENINGS AND MILL OF COLORADO FUEL AND IRON CO. AT WAGON WHEEL GAP, COLORADO.

The fluorspar vein crops out approximately in a line from the mill to the highest opening on top of the hill. A hot spring flows into Goose Creek just below the mill and another issues near the bathhouse, the large white building in the left foreground.

The vein zone has been traced from Goose Creek eastward up and along the ridge for a distance of more than ½ mile. The hill rises with a fairly steep slope to a height of about 700 ft. above the creek, and four tunnels and a sublevel tunnel have been driven on the strike of the vein zone to distances varying from 200 to 1400 ft. The thickness of the fluorspar varies from a few inches to 35 ft. The fluorspar extends from the crest of the ridge at least down to creek level some 700 ft. vertically, but appears to become thinner with depth, and no information was available as to whether it extends below creek level or not.

The vein zone extends eastward from the creek close to the point where the hot springs emerge but has not been identified west of the creek. The hot springs evidently bear some relation to the fissure vein and the travertine deposited by one of the springs contained 0.22 per cent of fluorine, but no fluorine was shown to be present in the waters issuing from the springs, according to the analyses given by Emmons and Larsen.(8)

The Wagon Wheel Gap deposits of fluorspar are owned by the Colorado Fuel and Iron Co. and spar has been mined from them since 1911. The product of the mine is treated in a mill (Fig. 3) at the foot of the hill and the product is carried by a tram road one mile down the creek and loaded on railroad cars at Wagon Wheel Gap station. The grade of spar containing 80 to 85 per cent calcium fluoride suitable for open-hearth flux has been used chiefly at the Minnequa works of the company, while higher grades, carrying at least 96 per cent calcium fluoride, have been shipped chiefly to East St. Louis, Ill., for use in the manufacture of hydrofluoric acid. In 1926 it was believed that less than 50 per cent of the known content of the vein had been mined out. Much good lump spar might be obtained here but has not been recovered because practically all the output is sent to the mill. Analyses of special mill samples of spar made by the Colorado Fuel and Iron Co. are shown in Table 1.

TABLE 1.—*Analyses of Mill Samples of Spar, Colorado Fuel and Iron Co.*

	CaF_2	SiO_2	Al_2O_3 and Fe_2O_3	$CaCO_3$	$BaSO_4$
Coarse	93.95	1.12	0.96	1.12	
Fines	95.40	1.15	1.00	1.15	
No. 1 and No. 2 jigs	94.93	1.08	1.60	1.05	1.65
High-grade jig bin	89.07	4.45	2.80	1.40	2.15

Jamestown District

In the mining district about Jamestown, Boulder County, fluorspar is a common vein mineral, in places constituting almost the whole vein filling. Other minerals that have been mined here are gold, silver and lead. Mines at which fluorspar has been the principal product consist of openings on Blue Jay Hill in the southeastern part of the district, and northwest of the village, particularly on the slope of Porphyry Mountain. The mines northwest of the village are the Wano, the Yellow Girl, the Brown Spar, the Invincible, the Chancellor, the Terry, the Alice, the Burlington, the Emmett, the Argo, and the Evans, all within an area of about one square mile. Jamestown fluorspar can generally be recognized by its characteristically dark purple color, but on weathering the color fades.

On Blue Jay Hill are well defined veins of fluorspar ranging from a few inches to 3 or 4 ft. thick, generally nearly vertical but striking in various directions. The spar is mostly coarsely crystalline, dark purplish in color, and carries in its fractures more or less siliceous impurity, some iron pyrites, and a little galena. Considerable lump ore has been shipped from here for flux at the open-hearth steel plant of the Colorado Fuel and Iron Co. at Pueblo.

At the extreme northwest end of the district a very distinct vein of fluorspar has been opened at the Evans mine. This vein cuts porphyritic rock, strikes northwest-southeast, and pitches steeply toward the northeast. It has been opened by surface cuts and a shallow tunnel for a distance of 100 ft. or more is about 2 ft. thick and consists of coarsely crystalline to glassy dark purple spar containing a little iron pyrites. It is reported that a shipment of lump spar to the steel plant in 1927 carried 89.56 per cent CaF_2 and 6.20 per cent SiO_2.

At the other mines mentioned the deposits vary from well defined veins to masses of country rock more or less replaced by finely granular fluorite. In some places there is only enough fluorite in the "porphyry" to give it a faint purplish hue, but in others there are masses of rock so far replaced by fluorite as to constitute a low-grade lump ore for open-hearth flux. There is more or less pyrite in this type of ore, however, which makes careful hand selection necessary, and emphasizes the need for an efficient method of concentrating the ore.

Fluorspar used to be hauled from the Jamestown district by team and wagon to Boulder over mountain roads. Now there is an excellent hard-surfaced road downgrade practically all the way, via James and Left-hand creeks to the mountain front and thence south to Boulder, and haulage of ore by motor trucks, a distance of about 15 miles, mostly downgrade, presents no difficulties.

A market for spar of steelmaking grades at current prices at the Colorado Fuel and Iron Co. open-hearth works seems fairly assured if a steady supply of spar can be furnished by the miners. To bring this about an effective method of cleaning and concentrating the spar must be perfected. Private efforts have been made along this line at a small mill in Jamestown, designed to handle the low-grade, disseminated spar of this district, but the results have not yet proved entirely satisfactory according to local reports. There is obviously an opportunity for study of the problem of the beneficiation of this particular type of ore by state or government laboratories. The Jamestown lean ore differs materially from those of the other districts of Colorado as well as from those of the Illinois-Kentucky district, in carrying principally siliceous impurities instead of calcite and barite. The following analyses of fluorspar from the Jamestown district made by the Colorado Fuel and Iron Co. show the proportions of impurities present:

CaF_2	SiO_2	Al_2O_3-Fe_2O_3	CaF_2	SiO_2	Al_2O_3-Fe_2O_3
76.05	19.8	4.2	79.06	15.24	5.26
83.76	12.2	4.0	86.75	.608	4.46
85.9	10.5	3.75			

North Gate

A deposit of fluorspar that has contributed the third largest production in Colorado occurs 3½ miles northeast of North Gate on the Laramie, North Park, and Western Railroad.

The deposit is in the form of a single vein in places and in others consists of sheets of vein material separated by country rock. The inclosing rock, where not so badly altered as to be unrecognizable, is a light pink, coarse-grained granite, weathering reddish. The granite contains also dikes of pegmatite and veins of quartz. The fluorspar extends along the west slope of a hill on the east side of a gulch opening from the north into the valley of Pinkham Creek and over the top of a low spur extending from the flanks of the Medicine Bow Mountains. The altitude at the portal of the 200-ft. level of the mine is about 8500 ft. The granite is much jointed in the vicinity of the deposit and considerably altered along the walls of the vein zone. Movement of the rocks has occurred both vertically and on the strike of the vein, as shown by slickensides within the mine.

The general strike of the vein is N.14°W. and the dip is generally very steep toward the west. In places the vein disappears entirely, but where present it varies in thickness from a few inches to 2 to 5 ft. of well defined vein material, and in places the vein zone may be 10 to 30 ft. across. In the main workings the vein had been followed for 875 ft., and north and south of these workings surface trenches, shafts and float have shown the presence of fluorspar for about a mile, although it may not be continuous for this distance. Vertically there is not much variation in thickness of the vein. (See Fig. 4.)

The North Gate fluorspar is generally in finely crystalline radial masses or in ribbonlike forms. Mammillary masses on the wall rock are common. Many of the radial masses, or "niggerheads," are found to have a core of granite. Intergrown with the spar are areas of hard, white and grayish siliceous material similar in character and relations to that in the spar at Wagon Wheel Gap. In some specimens this siliceous material grades into good spar but in places the separation is sharply defined. Iron pyrites in fine crystals is found occasionally, interlayered with fluorspar, and in places some of the pyrite is cupriferous. A small percentage of barite is present, and pockets of white claylike material are not uncommon. The common colors of the spar are light green, white and amethyst.

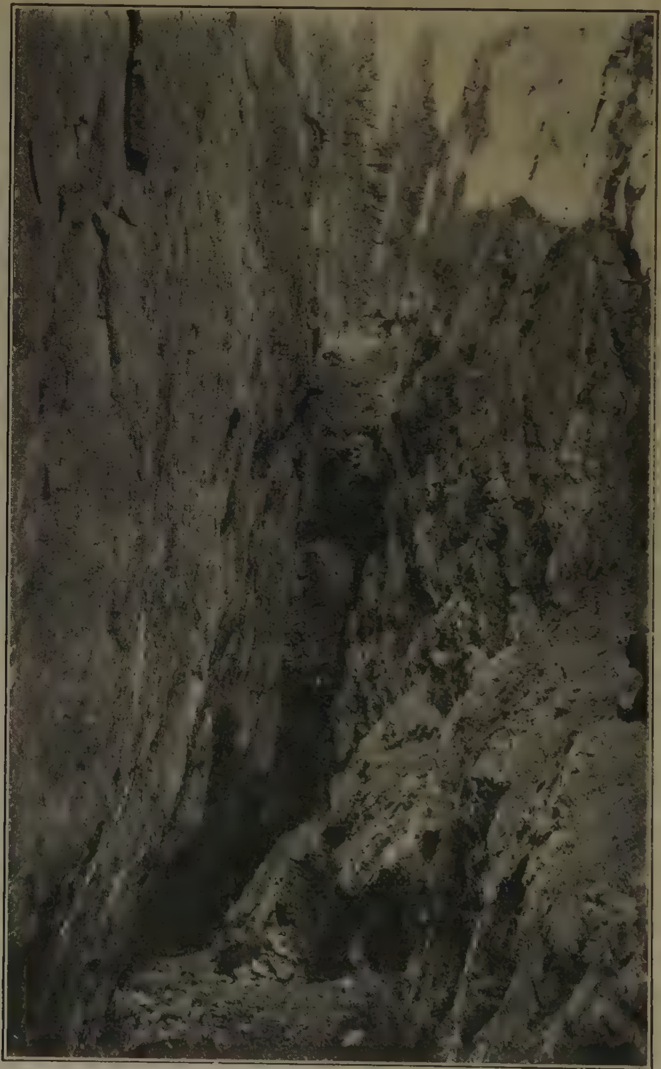

FIG. 4.—OPEN CUT ON NEARLY VERTICAL FLUORSPAR VEIN AT MINE OF COLORADO FLUORSPAR CORPORATION, NORTH GATE, COLORADO.

The mine was operated from 1922 to 1926. It was equipped with a mill having a capacity of 1000 tons per month, situated on the hillside at the mouth of the 200-ft. level. A high grade of open-hearth fluorspar was being shipped to the Illinois Steel Co. Analyses by this company of two 50-ton carloads showed respectively 92.86 per cent and 92.26 per cent calcium fluoride and 4.87 per cent and 4.16 per cent silica. The spar was hauled in 4-ton loads by motor truck to the railroad at North Gate, 3½ miles distant, all downgrade. Two to five truckloads a day were being shipped. On account of the long cold season and heavy snows, the mine and mill cannot be operated in winter.

IDAHO

Reports have come to the Geological Survey of discoveries of deposits of fluorspar near Stanley, Idaho. Mining men at Boise who are familiar with the Stanley region have stated to the writer that the Lucky Strike mine contains crystalline fluorspar in streaks ½ in. thick on the margins of dikes of porphyry. Another report was to the effect that crystalline fluorspar occurs as scattered veinlets up to 2 or 3 in. thick, cutting granite. Above Clayton in Spar Canyon thin veins and float fragments yielded "spar" for flux in a now abandoned smelter at Clayton, but it is not certain whether this was fluorspar or calc spar.

Notes by J. B. Umpleby, who spent several seasons in a geological reconnaissance of central Idaho, mention large quantities of fluorite occurring in association with quartz in a system of fissures on Little Casino Creek a mile or so above Salmon River. A little gold and silver was present in the veins, but owing to the remoteness of the locality, unless mining of these metals should be developed on an important scale, there is little possibility of recovery of the fluorite.

C. P. Ross has kindly supplied the following notes:

> In and near the Parker Mountain mining district on the southwestern border of Lemhi County, Idaho, small gash veins of coarse fluorite and crypocrystalline quartz cut the Challis volcanics (Oligocene?). The small size and sparse distribution of these veins precludes their being of commercial importance at present. The district contains gold-silver lodes but the fluorite veins are not known to contain precious metals.
>
> Fluorite occurs in a number of the auriferous lodes in the vicinity of Edwardsburg, in the Eastern Division of the Idaho National Forest, Idaho. So far as known, it is much more abundant in the Independence mine than in any of the others. This property is on Upper Smith Creek and is crossed by the Forest Service highway. The lodes therein are in the Yellowjacket formation (Algonkian) and are formed by silicification of the schistose quartzite and injection of vein quartz along joint and other openings. The principal lode is several thousand feet long and has a width of fully 200 ft. The fluorite is disseminated in coarse grains in irregular quartz veins, a yard and more in width which occur at intervals through the lode. The property has not been developed beyond some small prospect tunnels. It is held for its low-grade gold and silver ore and there may locally be enough fluorite to be of value as a by-product if the mine is developed.

NEVADA

Certain fluorspar claims near Beatty, Nev., have been described by Ladoo,(10,11,12) and in order that the data on the Western States shall not lack information on Nevada, some notes are given herewith.

As described by Ladoo there are 22 fluorspar claims in Nye County, of which 9, including the Daisy group and the Fluorite, Blue Bird, and Yellow Spar claims, are on the northeast end of the Bare Mountains, 4½ to 6 miles southeast of Beatty, the shipping point on the Tonopah and Tidewater Railroad. The remaining 13 claims are known as the Diamond Queen group and are 7 miles over the mountain southeast of the Daisy Group. The Diamond Queen Group is about 11½ miles over the mountain southeast of Beatty, but is about 22 miles distant by the only practical haulage road.

The geologic formation containing the fluorspar is a dark gray limestone (Ordovician?) faulted in many directions and intruded by rhyolite. The fluorspar occurs as a vein filling in the fault fissures, as a cement in brecciated limestone, and as a partial replacement of breccia and wall rock. The thickness of the fluorspar filling in the veins generally ranges from 10 in. to 4 ft., but there are places where the deposits attain thicknesses of 10, 17 and 20 ft. At the Fluorite claim the vein filling is considered not to contain more than 50 per cent of fluorspar.

The fluorspar is largely of a soft, finely granular type, purple in color, but there is some yellowish lump and boulder material. Samples of spar from the Daisy and Blue Bird claims showed respectively the following percentages:

	DAISY	BLUE BIRD
Calcium fluoride (CaF_2)	75.31	91.22
Calcium carbonate ($CaCO_3$)	19.84	1.37
Silica (SiO_2)	4.15	6.92

Developments consist of several prospects, tunnels, shafts and drifts. The shafts reached depths of 48, 75 and 160 ft. There is no water at the mines and the ore had to be hauled to Beatty where a concentrating and grinding mill had been built and had been in experimental operation in May, 1920. The principal problem in milling was to recover the very finely granular fluorspar in sizes as coarse as possible, and to this end it was proposed to run the entire product over tables in order to render the coarse gravel suitable for flux and the finer grades suitable for the glass and enameling trades.

More than 100 carloads of fluorspar are reported to have been shipped from these deposits near Beatty. It seems reasonable to consider that there are reserves many times as large, which may be drawn upon ultimately if methods of beneficiation are perfected. Ladoo, however, did not view the situation optimistically in his reports, the objects of which were more concerned with the immediate outlook.

NEW MEXICO

The data on fluorspar deposits in New Mexico are summarized largely from a bulletin(9) prepared by William Drumm Johnston, Jr., for the New Mexico Bureau of Mines and Mineral Resources in 1928 and from earlier notes of the writer on deposits in Grant and Luna Counties.

Johnston examined some 40 deposits of fluorspar and comments as follows:

> The fluorspar resources of New Mexico are varied. Large deposits of high-grade ore comparing favorably with those of the Kentucky-Illinois field are rare. There are, however, numerous small deposits scattered over the southwestern part of the State, each of which can be operated by a few men, and whose combined production will attain a substantial figure. In addition the large siliceous deposits of the Sierra de los Caballos are producing high-grade milled fluorspar, and additional development of these ores is in progress.

Two types of fluorspar deposits have been recognized in New Mexico; *viz.*, vein fillings in igneous and sedimentary rocks, and blanket replacements in limestone. The latter are of minor importance. The greatest depths to which fluorspar veins have been followed are 286 ft. at the Tortugas mine and 320 ft. at the Galena King mine, and no change in tenor of the ore was noted at those depths. Since the fluorspar veins are simple fissure fillings, their shape and extent is primarily determined by the shape and extent of the preexisting fractures, and there is no reason to expect a consistent pinching of such fissures at depths less than 1000 ft.; mineralization can therefore be expected to be continuous to that depth at least.

In places the veins show conspicuous lateral variation from fluorite to barite and this variation may occur vertically also. Much secondary quartz accompanies the replacement of limestone by fluorspar, the quartz forming a narrow band advancing into the limestone just ahead of the fluorite. The outcrops of fluorspar are usually inconspicuous, but where the ores are siliceous the vein may stand as a dikelike ridge as much as 20 ft. above the weathered country rock. On the outcrop the fluorspar appears white because the delicate colors characteristic of unweathered spar are quickly bleached by sunlight. Deposits are often found through tracing to the outcrop loose fragments of spar on hillsides and in ravines.

Deposits of fluorspar have been described in the Sierra de Los Caballos, in Cooks Range, in the Tonuco, Organ, Little Florida, Telegraph, Pinos Altos, Little Burro, Mogollon, Zuni, Sandia, Manzano, San Andreas, and Oscura Mountains; in the Sierra Ladrones, and in the Joyita and Little Burro Hills; but only a few of the most important groups of deposits will be described here.

Sierra de los Caballos

The fluorspar deposits near Hot Springs are on the western scarp of the northern end of the Sierra de los Caballos at about 1000 ft. above the Rio Grande at Hot Springs. They are reached by a good road, which crosses the Rio Grande at Hot Springs and goes upgrade 4½ miles to the property. In times of low water the river may be forded, otherwise a ferry is used. The fluorspar veins crop out at the base of the steep face of the mountain and at the heads of westward-draining ravines. The country rock cut by the veins is limestone of the Magdalena formation. The ore occurs in veins striking northwest-southeast, three of which have been opened on several levels on the hillside. The thickness of the veins ranges from 5 to 15 ft. of workable ore. One vein has been traced a distance of 2000 ft. up the hillside by means of float. In places it is a well marked single fissure filling but elsewhere it consists of a sheeted zone as much as 60 ft. wide containing much silica and calcite. These fluorspar deposits are notably siliceous. Fluorite and quartz were the primary vein filling, with considerable replacement of the wall rock.

A mill consisting of two parts, a dry plant at the mines and a wet plant at Hot Springs, has yielded a product carrying more than 97 per cent calcium fluoride. The finished product is hauled by truck 21 miles to Engle, on the Atchison, Topeka and Santa Fe Railroad. Some ground spar has been shipped for use in glass and hydrofluoric acid manufacture. Considerable waste containing fluorspar results from the milling. Samples of concentrates from the dry mill and from the wet mill gave the following percentages:

	Dry Mill	Wet Mill
Calcium fluoride (CaF_2)	95.80	98.13
Calcium carbonate ($CaCO_3$)	1.48	1.00
Silica (SiO_2)	2.61	0.64

Johnston believes that the existence of a large quantity of fluorspar in the Hot Springs district is certain but that the treatment of the siliceous ore presents many difficulties and there is need for a more satisfactory milling procedure in order to render these deposits available to the industries of the West. The fact that concentrated ground spar is the only present product seriously limits the market for the output.

As to fluorspar reserves, Johnston's estimate of the probable ore occurring in the Universal, Oakland and White Star veins within a vertical distance of 200 ft. below the lowest outcrop of each vein, based on the assumption that they retain their outcrop width to that depth, is in the neighborhood of 300,000 tons. He believes that recovery of this tonnage, while possible, may not yet be economic under present conditions.

The Lida K claims are on the west face of the southern end of the Sierra de los Caballos, 23 miles by road from Hot Springs, and about 450 ft. higher than the Rio Grande. The shipping point is Hatch, 20 miles to the south.

The fluorspar, associated with quartz and a small proportion of galena, fills a nearly vertical fissure 3 to 5 ft. wide in pre-Cambrian muscovite schist and granite gneiss, and is traceable for 4000 ft. At the shaft the vein material contains generally between 72 and 77 per cent calcium fluoride, between 15 and 25 per cent silica, 0.6 to 1.6 per cent carbonate, and a trace to 0.6 per cent galena, although in places as much as 3 to 7 per cent. The siliceous nature of the fluorspar will necessitate its being properly concentrated, and a special process has been worked out and tried experimentally, but marketable material had not been produced in quantity up to 1928, nor had any ore been shipped.

Organ Mountains

The Tortugas mine is on Tortugas Mountain, in Dona Ana County, 5 miles northwest of Mesilla Park, on the Atchison, Topeka and Santa Fe Railroad.

The fluorspar occurs chiefly as a filling of fissures in Paleozoic limestone with replacement of secondary importance. The veins contain fluorite as the principal filling of the fissures. Two veins, the Tortugas and the Jones, have been opened in mining. The Tortugas vein is exposed in workings for a distance of 1000 ft. along the strike and can be followed by surface residual spar for a total distance of 1400 ft. In August, 1927, it had been opened to a depth of 286 ft. without change in character. The thickness of the vein varies from 2 to 8 ft. and it pinches out at the extremities of the outcrop. The Jones vein is a fracture zone intersecting the Tortugas vein north of the workings on that vein. The total thickness of the fracture zone is more than 15 ft. and the fluorite occurs as fillings and replacements in the zone of shattering. The ore is bottle green in color and occurs in large crystals, some of which reach 3 in. across. Silica and calcite occur with the fluorspar, but barite is rare.

The Tortugas mine was opened in 1920 and after a period of idleness was reopened and was active in 1927. To 1927 the total officially recorded production was a little more than 15,000 tons. The spar shipped varied from low-grade fluxing material carrying 80.91 per cent calcium fluoride, 13.15 per cent calcium carbonate, and 5.64 per cent silica, to ground spar for ceramic use carrying respectively 93.76 per cent, 2.50 per cent, and 3.42 per cent of the same substances. Mining was done by overhead stoping and ore is chuted down to the 286-ft. level, from which it is raised to the surface by a gasoline hoist. At the surface the acid lump spar is sorted out by hand. From the upper platform it is carried 300 ft. by aerial tramway to the east foot of Tortugas Mountain, where it is again picked and the mine run crushed and loaded on trucks and hauled 4 miles to the mill near Mesilla Park.

Johnston considers that the recoverable fluorspar in the Tortugas vein remaining in the mine above the 286-ft. level amounts to 9000 tons, and that an additional depth of 200 ft. on the vein is practical mining ground, so that if the spar persists to that depth in its present tenor, an additional 13,500 tons of spar can be expected, making the total reserve of the Tortugas vein about 22,500 tons of recoverable fluorite, which, at the present rate of production would afford ore reserves for a number of years. In addition to the Tortugas mine there are in the Organ Mountains, the Hayner and Bishop's Cap prospects, where there are apparently favorable conditions for seeking valuable deposits of fluorspar.

Cook's Range and Little Florida Mountains

The following fluorspar mines and prospects are situated in Cook's Range and vicinity: the White Eagle mine in Grant County at the north end of Cook's Range; the Diamond Prospect, about 3 miles north of the White Eagle; the Sadler mine, on Fluorite Ridge, an outlier of Cook's Range about 11 miles northeast of Deming; the Cox Prospect, near the crest of Fluorite Ridge, and in the little Florida Mountains the Duryea claims, 14 miles southeast of Deming.

The Sadler mine has had the longest history and the most extensive development. The ore occurs in veins cutting monzonite porphyry, and as a fissure filling in monzonite porphyry and volcanic agglomerate, and is associated with basaltic dikes. The mine was visited in 1910 by the writer when mining was in progress and it continued to produce a good grade of fluxing spar for some years, but when visited by Johnston in 1927 it was idle and practically all visible spar having a thickness of more than 2 ft. had been mined so that no further development seemed justified unless the price of metallurgical spar should advance materially.

Telegraph Mountains

The Great Eagle mine is in Grant County, 32 miles north of Lordsburg, the nearest railroad point, on the Southern Pacific Railroad. It is reached by an ungraded level road across the bolson plain to the arroyos of Gila River, where for a distance of 9 miles there are a number of steep grades where the road crosses arroyos and gravel spurs. The property was examined by Burchard in 1926 and by Johnston in 1927.

The country rock is pink and gray, coarse granite, probably of pre-Cambrian age, intruded by stocks and sills of rhyolite and diabase, probably of Tertiary age. A fault zone, 100 ft. wide, that strikes N.40° W. across the western end of the granite area, contains fluorspar as a filling of fissures within the zone. The fissures vary in thickness, reaching a maximum of 25 ft. Not all of this is fluorspar, for it is estimated that the 25-ft. face yielded with hand sorting about 40 per cent of metallurgical lump spar, the remainder of the vein consisting of partings 1½ to 3 ft. thick of material too siliceous for use as flux.

The total exposed length of the vein on the surface is 500 ft. and there were extensive underground as well as surface openings made in 1917–1921, as a result of which about 3000 tons of metallurgical lump spar was shipped. In 1919 a mill was erected but it did not prove satisfactory and after having produced a few tons of ground spar it was dismantled and removed. The difficulty in treating this fluorspar is the presence of much spar in which there is an intimate mixture of siliceous material. A sample representing the best ore available taken by Johnston contained the following percentages: calcium fluoride, 92.97; calcium carbonate, 0.78, silica, 2.92; alumina and ferric oxide, 3.30. The average fluorspar content of 1983 tons of ore shipped between July 1, 1918, and January 1, 1919, is said to have been 91.5 per cent.

General Conclusions Regarding Fluorspar in New Mexico

The published notes on mines and prospects indicate that New Mexico possesses large reserves of fluorspar. Estimates of available tonnages will vary with the bases used for the probable dimensions or extent of the deposits. Mr. Johnston is more optimistic than the present writer as to the persistence of the ore to depths considerably greater than at present demonstrated.

The total production of fluorspar in New Mexico as recorded by the U. S. Bureau of Mines, including the year 1931, is 56,261 short tons, and a rough, liberal estimate by the writer of the reserves of concentrates of commercial metallurgical grade fluorspar is about 400,000 short tons.

Localities indicated by Johnston as favorable for prospecting exist in all the ranges where deposits of fluorspar have already been found and he points out that some fluorspar, because of its resemblance to the minerals quartz and calcite have doubtless been overlooked by those who sought metalliferous deposits. The writer recalls that a large deposit of fluorspar in Colorado was not recognized for years on account of its resemblance to amethystine quartz, with which the local miners were familiar.

UTAH

According to Heikes(17) the only deposit of fluorite in Utah, from which the mineral has been shipped in commercial quantity, is at the north end of Wild Cat Mountains, in Tooele County, 20 miles southwest of Clive, a station 74 miles west of Salt Lake City on the Western Pacific Railroad.

Locally the Wild Cat Mountains consist of limestone, shale and quartzite, cut by a few dikes of basalt. The fluorspar occurs in veins in fissures that cut the limestone but show little displacement. The veins range from a few inches to several feet in thickness, and crop out at various places over a 90-acre tract. Where the vein walls have been broken by numerous fractures the limestone has been considerably replaced by fluorite, and in such places the vein may have a workable width of 15 ft. or more. Other minerals are barite, pyrite, chalcopyrite, and traces of gold and silver.

The property was operated in 1920 by T. G. Jensen, of Salt Lake City. Notes by Clark and Meacham indicate that the mine was best reached by a 20-mile, practically level, road across the desert, which was impassable several months in the year. Hauling was done with a 1½-ton truck. The deepest workings were about 50 ft., where salt water was encountered. The fluorspar occurs only in pockets, which usually contain 15 to 30 tons of high-grade, crystalline spar. Eventually the long haul, the low prices since the war, and the diminishing quantity of easily available fluorspar made it unprofitable to continue operations. More spar could be produced under conditions of strong demand and high prices, but no estimate of reserves can be made. Analyses show the following percentages:

CaF_2	$CaCO_3$	SiO_2
97.23	1.36	0.48
89.15	6.55	3.05
86.14	9.60	3.16
85.46	11.10	2.4

Fluorspar having been reported from near Ogden, a visit was made to the locality in question by the writer in company with Prof. J. G. Lind, of Weber College, Ogden. Several veins 4 to 12 in. thick containing fluorspar were seen on a spur of the west slope of the Wasatch Mountains, 650 to 930 ft. above the alluvial apron at the base of the mountains about 2¼ miles northeast of Ogden. Where veins intersect there may be a thickness of 2 ft. of spar. The rocks associated with the fluorspar are pre-Cambrian gneiss intruded by masses of diorite and cut by dikes of pegmatite in which streaks of fluorite are found. A few small open cuts were noted and a crosscut tunnel 240 ft. long N.80° E., which intersects several of the veinlets. The best fluorite was in mammillary incrustations, 3 or 4 in. thick, with banded structure on the hanging-wall side of the veins. Pieces of spar of this sort have broken off from the outcrops and have weathered into rounded fragments of float below. The fluorspar is associated with white, fine-grained, siliceous material, as in other deposits in the West. These deposits cannot be considered of commercial importance. If larger deposits should be found here the location would favor their development.

WASHINGTON

A deposit of fluorspar 6 miles northwest of Keller, Ferry County, Wash., near the middle of sec. 24, T.30, R.32 W., was examined by the writer in August, 1926. It is in mountainous country on a pine-wooded slope in the drainage of Jack Creek, which flows into San Poil Creek, a northern tributary of Columbia River (Fig. 5). The deposit is about 540 ft. higher than the village of Keller. It is on the east slope of a hill which rises about 400 ft. above the adjacent valley but the surrounding hills and mountains show much greater relief. The haul to the Republic-Wilbur road in San Poil Valley would be downgrade about three miles.

The adjacent rock is medium coarse grained pink granite with greenish areas. The fluorspar occurs as a well defined vein striking N.72° to N.78°W. The vein is nearly vertical but in places dips steeply toward the south. For most of its exposed length the vein is single but in several places it branches into two parts or else is divided by sheets of rock or barren material. A dikelike mass of light gray fine-grained silicified rock and quartz veins both cut the tunnel walls. Fluorite is the principal vein mineral. The vein varies in thickness from 3 to 4 in. to about 3 ft. Its vertical and longitudinal limits have not been determined, but extend several hundred feet according to evidence of prospects, which consist of several pits and two tunnels. In the tunnels the proportion of spar to material removed is less than 50 per cent. Where the vein branches or splits the included rock is in places granite and in others a dense, siliceous rock.

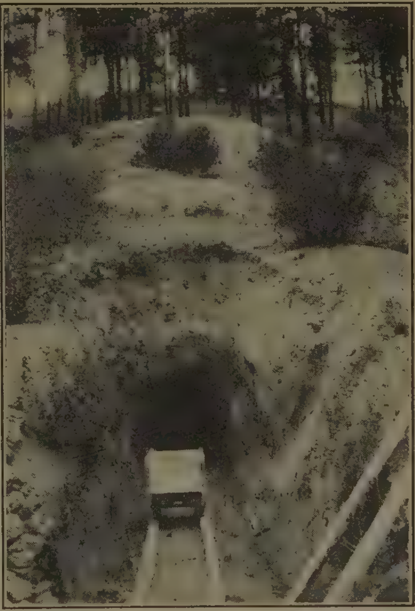

FIG. 5.—ENTRANCE TO LOWER TUNNEL OF MITCHEM FLUORSPAR MINE, 6 MILES NORTHWEST OF KELLER, WASHINGTON. Upper dump is from a higher tunnel, showing that this vein is nearly vertical.

The fluorspar is largely of an unusual purity, consisting principally of light green crystalline material, although opaque white (siliceous), pale purple, and clear colorless spar occur. A very little pyrite was seen in the spar in places, usually in the purplish tinted variety.

One tunnel was driven by the owners of the ore smelter at Trail, B. C., and is reported to have produced about 60 tons of high-grade lump spar in 1918 for making hydrofluoric acid. Evidently the production was hand-picked spar. The spar that was shipped was hauled by truck about 15 miles to Columbia River, on which it was carried by stern-wheel steamboat to Pateros, transferred to the Great Northern Railroad and carried by rail to Trail, B. C. The total cost of the spar thus transported was said to have been very high and except in wartime was prohibitive. The royalty paid the owner, H. C. Mitchem, of Spokane, was $5 per ton. The Trail smelter is now supplied with fluorspar from the Rock Candy vein, 2 miles north of Grand Forks, B. C., and not far from the Canadian Pacific Railroad. The Rock Candy vein is said to be larger than the Keller deposit and to contain spar of similar character.

Unfavorable conditions affecting mining here are the isolation of the place, the poor transportation facilities, and the distance from labor supply and markets. If a mill were to be built, water would have to be pumped from Jack Creek, distant about three-quarters of a mile. If a road were built to the highway in San Poil Creek valley, spar might be hauled by truck to Republic, thence by rail to Trail, probably cheaper than the way it was transported in 1916.

The reserve of fluorspar estimated for this deposit does not seem large enough to warrant expenditure of a large sum in developments, mill, or road building. The most practicable development probably would be to proceed in a moderate way to produce picked lump spar and to improve the road enough for truck haulage to the San Poil valley highway.

Comment

The tables of production and estimates of reserves of fluorspar in the Western States disclose widespread distribution of deposits of this useful mineral. As indicated in the text, fluorspar occurs also in states from which no production has been reported, such as Idaho; fluorspar has also been reported in the Black Hills of South Dakota, and there are probably occurrences in the remaining Western States of Wyoming, Montana and Oregon. The deposits described in California are in only one locality but others in southern and central California have come to attention as well as many more in Arizona. Thus far the principal developments of fluorspar have been in fairly accessible places, and with the building of railroads to new mining districts and development of roads for truck haulage there will be discovery and development of deposits of fluorspar as yet unrecorded.

The estimates of reserves of fluorspar are necessarily rather rough as compared with estimates that may be made for a well developed district such as southern Illinois. No attempt has been made to distinguish between "proved," "probable," or "possible" ore reserves, but it is thought that the qualification "probable" is the most reasonable that might be applied to most of the deposits, although "possible" might be better for certain of the large deposits near Hot Springs, N. M. The estimates for the larger deposits in Colorado are based upon the most exact data and it seems interesting that New Mexico should show reserves comparable with those of Colorado. This is due to the large reserves of low-grade spar in the Hot Springs region, the production of which will involve concentration on a large scale and will require much improvement over present methods of concentration, so that the figures do not really give a direct comparison between the availability of the reserves in the two

TABLE 2.—*Production and Probable Reserves of Fluorspar, Western United States*

(Subject to Revision)

State	Deposits	Distance to Shipping Point, Miles	Total Production, Including 1931, Short Tons	Estimated Probable Reserves of All Grades of Spar Mostly Fluxing Grade, Short Tons
Arizona	Safford et al., Greenlee Co.	8 to 16 N.E. Duncan	1,742	90,000
	Castle Dome district, Yuma Co.	25 N.NE. Dome		
	Pima Co.	—(?)		
	Yavapai Co.	—(?)		
California	Several groups of claims	3 S.E. Afton	—	75,000
Colorado	Colorado Fluorspar Corp. Jackson Co.	3½ N.E. North Gate	170,724	400,000
	Jamestown district, Boulder Co.	16 N.W. Boulder		
	Duffields Cather Springs, El Paso Co.	On Railroad		
	Barstow, Ouray Co.	On Railroad		
	Antelope Cr., Custer Co.	16 S.E. Westcliff		
	Jefferson, Park Co.	4 N. Jefferson		
	Evergreen, Jefferson Co.	12 W. Morrison		
	Wagon Wheel Gap Mineral Co.	1¼, Wagon Wheel Gap		
	Browns Canyon, Chaffee Co.	3 D. & R. G. R. R.		
New Mexico	Fluorspar Mines of America Co., Sierra Co.	5 S.E. Hot Springs	56,261	400,000
	Southwestern Fluorspar Corp., Lida K mine, Dona Ana Co.	20 N. Hatch		
	Hayner Prospect, Dona Ana Co.	4½ S. Organ		
	Bishop's Cap, Dona Ana Co.	10 E. of Mesquite		
	Tonuco, Dona Ana Co.	1 E. Heathden		
	Tortugas, Dona Ana Co.	5 N.E. Mesilla Park		
	Great Eagle, Grant Co.	32.5 N. Lordsburg		
	Gila, Grant Co.	30 N.W. Tyrone		
	Fluorite Ridge, Luna Co.	6 N.W. Mirage		
	Cook's Range, Luna Co.	17 N.E. Spaulding		
	Nackaye, Sierra Co.	20 N.W. Hatch		
Nevada	Daisy and Diamond Queen Groups, Nye Co.	6 S.E. Beatty	5,077	70,000
Utah	Silver Queen, Tooele Co.	20 S.W. Clive		
Washington	Keller, Ferry Co.	28 N. Wilbur		
		Totals	233,804	1,035,000

SUMMARY

	Recorded Production, Short Tons, Through 1931	Estimated Reserves, Short Tons
Arizona	1,742	90,000
California	......	75,000
Colorado	170,724	400,000
New Mexico	56,261	400,000
Nevada Utah Washington	5,077	70,000
	233,804	1,035,000

states. In Colorado certain of the deposits are large and of fairly high grade, their market is assured, and so they will continue production more rapidly than any in New Mexico at present.

The question of markets for fluorspar in the West is of course vital to the search for and development of deposits of the mineral. At times western fluorspar of acidmaking grade is shipped as far east as Brooklyn, and enameling and glass grades have gone to Philadelphia, Chicago, Muncie, Ind., and St. Louis and some goes to the Pacific Coast. A little fluxing spar has gone to Duluth, Minn., Gary, Ind., Kansas City, Mo., and other mid-west steel plants, but the greater part of the production is consumed at the open-hearth steel works at Pueblo, Colo., and this consumption is limited, of course. There are open-hearth steelworks near Seattle, Wash., and San Francisco and Los Angeles, Calif. These plants use some fluorspar from western mines but some is also imported from China and England. Imports from European countries probably always will be a menace and will retard the use of domestic spar as a flux at Pacific Coast plants except from deposits within a moderate rail haul.

In estimating the reserve tonnage little consideration has been given to the matter of grades and to the question of price. The tonnage is estimated for the most part on the basis of metallurgical fluxing grade, but where high-grade crystalline lump spar occurs in abundance this fact has been mentioned. Of course, high prices, such as prevailed during wartimes, and cessation of imports might bring out reserves as yet unknown, but it is believed that many of the deposits mentioned herein which are susceptible of greater development would fulfil the demands for western fluorspar through another emergency. Compared with reserves of fluorspar in the Kentucky-Illinois district, those in the Western States are not an important factor in the steel trade east of the Rocky Mountain region, but their influence may be felt to some extent in the glass and enameling fields.

The total output of fluorspar recorded for the Western States to the end of 1931, that is, 233,804 short tons, is about 7 per cent of the total of 3,340,266 short tons for the United States. The average annual output of the Western States for the last 10 years, about 9400 short tons, is about 8 per cent of that for all the states, and the average annual production of the Western States for the last 5 years, about 7400 short tons, is only about 6.7 per cent of the average total domestic production, which indicates that the rank of the Western States may now be diminishing. At the present rate of production the estimated reserves of fluorspar in the Western States would appear to require about 140 years for their exhaustion, compared with less than 50 years for those of the Kentucky-Illinois district, but in view of the rapidly changing conditions to which industry is subject, such a comparison is of little significance.

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DISCUSSION

(H. Foster Bain presiding)

R. B. LADOO,* Newton, Mass. (written discussion).—I believe Mr. Burchard has been somewhat too optimistic as to the feasibility of profitable commercial production at many of the deposits, and too generous in his estimates of recoverable tonnages.

Nearly if not quite all of the western fluorspar deposits have a siliceous gangue, while the Illinois-Kentucky deposits (except for the natural "gravel" deposits) occur in limestone and have a calcareous gangue. The difference is very important from the standpoint of the value of the deposits as sources of fluorspar of metallurgical grade. Limestone is in itself a flux and as an impurity in fluorspar acts merely as a diluent. But silica consumes flux and every percentage of silica in fluorspar requires roughly 2.5 per cent of calcium fluoride to flux it. Thus there is a great difference in value between an 85 per cent gravel spar with limestone as the bulk of the remaining

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* R. B. Ladoo Co.

15 per cent and an 85 per cent spar in which the remaining 15 per cent is chiefly silica. If the spar contains 10 per cent silica the available calcium fluoride is not 90 per cent, but 90 per cent less 25 per cent (2½ times 10 per cent), or 65 per cent.

The significance of this fact is that, while considerable amounts of spar of glass and enamel grade can be sorted by hand from many of the western deposits, mechanical concentration of the ore to make spar of metallurgical grade usually results in a product high in silica. Mr. Burchard comments on this problem at the Great Eagle mine at Lordsburg, N. M., but it applies also to many of the other western deposits.

If we consider as "reserves" probable tonnages of calcium fluoride in the ground regardless of cost or feasibility of extraction, or of acceptability of the products made, the estimated tonnages tabulated are probably conservative. But if we are interested only in fluorspar that can be produced of a quality and in such physical condition as to be acceptable to the steel trade, and marketed at a profit in eastern markets, where the bulk of the spar is consumed, at a price that can reasonably be obtained even under war conditions, I would say that the estimates of reserves are much too generous. My views, of course, are based on the deposits as I saw them over 10 years ago, and I did not see all of the deposits that Mr. Burchard covered.

New and improved methods of concentration may be developed, standards of quality may be changed, transportation costs may be lowered; any number of things might happen, including new methods of making steel that would eliminate the need for fluorspar entirely. But in estimating ore reserves today I think we should include only ores that are commercially valuable under present conditions.

Other factors that are against the profitable and successful commercial recovery of fluorspar from many of the western deposits are listed below:

1. Great distance from a railroad and inaccessibility even for motor truck.
2. Narrow and irregular veins, resulting in high mining costs.
3. Friability of ore, resulting in a very fine, sandy concentrate, not acceptable to most eastern steel producers.
4. Presence of barite and other objectionable impurities.
5. Lack of water supply for concentration.
6. High freight costs to the important eastern steel markets.
7. Severe winters making mining and hauling costs high and conditions difficult.

Of course, not all of these factors apply to any one deposit.

Much of the recorded production from many of the western deposits came from war-time operation under the stimulus of high prices. Operators of several of the properties have admitted that their entire production was shipped at a loss. Many properties have changed hands repeatedly, each temporary owner shipping a few car loads at a loss and then quitting. Some of the ore shipped was rejected on quality and not paid for. A record of shipments over a period of years consequently is not good evidence of the commercial value of a deposit.

As to the possibility of undiscovered deposits, one thing has always impressed me. Under the stimulus of extremely high prices during the war no really important new fluorspar deposits were discovered in this country, in the west or elsewhere.

If we regard the western deposits solely as sources of metallurgical-grade spar for western steel mills only, the situation is somewhat more favorable. Western steel producers have not had as rigid specifications as eastern steel mills. But while high-grade imported spar is readily available on the Pacific Coast, western fluorspar producers will have difficulty in competing for the business of the Pacific steel mills. The Coloradol Fuel and Iron Co. has its own deposit at Wagon Wheel Gap, Colo., which always has been and apparently still is the largest and most consistent producer in the West.

My discussion has applied chiefly to metallurgical grade of fluorspar. Acid and enamel grades of spar occur in many of these western deposits and, as they command a higher price, it often pays to work some of these deposits on a small scale and sort out the high grade of spar by hand.

My general feeling about estimates of ore reserves such as this is that too much reliance is placed on geological evidence and not sufficient attention is paid to engineering, technological and economic considerations.

A. M. Smith,* Reno, Nev. (written discussion).—In addition to the fluorspar deposits near Beatty, Nev., which have been described by Mr. Burchard, there are three other deposits in Nevada, from each of which fluorspar has been mined and shipped. These appear to have considerable promise, and may become of considerable importance with the development of the fluorspar market on the west coast.

The first of these deposits, the Baxter mine, is on the line between Churchill and Mineral counties, 5 miles west of the silver camp of Broken Hills. The fluorite outcrops as a vein in Tertiary lavas for a length of some 2000 ft. The crystalline fluorite content of the vein is from 10 in. to 4 ft. wide, the strike is north, and the dip about 90° westerly. Along the outcrop are many cuts, pits and trenches and two shafts. The main shaft is an incline in the vein, about 100 ft. deep, with 100 ft. of drifting along the vein at the bottom. The ore averages 98.55 per cent calcium fluoride. One carload shipped in 1928 is reported to have analyzed 99 per cent calcium fluoride. About 800 tons was shipped to The Pacific Coast during 1929, and smaller amounts in 1930 and 1931. Large pure lump product is easily mined. The present disadvantage is a 75-mile desert-road truck haul to the Southern Pacific Railroad at Fallon. V. N. Baxter, of Broken Hills, Nev., is the owner.

The second deposit is in Mineral County, 3 miles south of Mt. Montgomery station on the Owens Valley Branch of the Southern Pacific Railroad, where a large deposit is owned by Sol Summerfield, of Mina, Nev. The mineral occurs in a series of parallel veins in Tertiary lavas in which rhyolite predominates, and is near a contact with limestone. There are four or more veins, which strike northerly across a canyon, which they cross at right angles. Three of these veins have been partly developed by tunnels. About 200 tons was mined and shipped to Los Angeles in 1929, since when the property has been idle. The deposit is very large, but is of low grade. Concentration by some method will be necessary to separate siliceous gangue material, which is the principal adulterant. An examination by the Nevada State Bureau of Mines in 1931 indicated a very large tonnage of fluorite of low grade, containing calcite and quartz. A report by a geologist for Southern Pacific Railroad Co. states that the tonnage will run into hundreds of thousands. Transportation costs to the Los Angeles markets will be moderate, as the mine is less than ½ mile from the railroad, and at a higher elevation. Presumably an aerial tramway would be the best way to effect the conveyance of the ore to loading bunkers on the railroad track.

Another fluorspar deposit is in Crescent mining district, Clark County, Nevada, on the Nevada-California boundary line; possibly a part of this deposit is in California. There are several veins of fluorite on the property, the largest being about 3 ft. wide and developed or explored by a shaft about 40 ft. deep. Some 600 ft. distant from this vein is another about 10 in. wide, and near by another having a width of about one foot or slightly more. With the exception of one of the narrow veins, which has yielded an 85 per cent fluorite, the ore is low grade, and presumably would require concentration to bring it up to a marketable product. Several tons are said to have been shipped about the year 1925. The mining is said to have been performed by Thomas Thorkelson, of Hollywood, Calif., who is associated with the American Borax Company.

It is certain that the fluorspar deposits in Nevada, if stimulated by a better Pacific Coast market, or by a steel and iron industry which may build up at Boulder Dam as a result of the vast amount of exceedingly cheap electric power that will be available there in four or five years from now, will become of much value to the nonmetallic industries of the western states.

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* Mining Engineer, Nevada State Bureau of Mines.

P. M. Tyler,* Washington, D. C.—Fluorspar is being floated in Southern Illinois by the Franklin Fluorspar Co., by methods developed by W. H. Coghill, at the U. S. Bureau of Mines Station at Rolla, Mo. In the light of this, the transportation of some of the western fluorspar may not be as remote as it would now seem, and Pueblo, Colo., may not be its only outlet. If a rate of $5 per ton to tidewater can be procured for the fluorspar concentrate, similar to that now prevailing on New Mexico potash, it may be feasible to ship the western fluorspar to eastern markets. The coastwise freight would be less than $2 in 1000-ton lots. It is now advantageous to import fluorspar as the Illinois and Kentucky production cannot compete east of Pittsburgh, Pa.

E. F. Burchard (written discussion).—Mr. Ladoo's discussion brings out some very important points for consideration. The distinction between the fluxing value of the Ohio Valley deposits having a calcareous gangue and of the siliceous western deposits is very real. Eventually, however, it may be possible, by flotation, as suggested by Mr. Tyler, to materially reduce the siliceous impurity and produce a concentrate carrying much more than the minimum requirement of calcium fluoride. Other factors enumerated by Mr. Ladoo that militate against profitable commercial recovery of spar from many western deposits are valid and will retard developments.

As to the possibility of undiscovered deposits, Mr. Smith's discussion indicates the existence of deposits heretofore not prominently recorded. The fact that during the period since 1909, when the writer first visited the Colorado and New Mexico fields, the bulk of the deposits now known were gradually discovered incidental to general prospecting may be more encouraging than if they had been discovered as the result of war-time stimulation. I do not think there was any great effort made to discover new deposits during the war. Manpower was limited and prospecting was not normally active.

Before the present paper was published, it was necessary to reduce its length by about 40 per cent, so that many deposits received no mention. Furthermore, the paper was not designed primarily as a discussion of commercial fluorspar reserves, but rather as an outline of the situation in its relation to industrial preparedness in the event of an emergency in which fluorspar might be regarded as a strategic mineral.

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* Chief Engineer, Rare Metals and Non-Metals Division, U. S. Bureau of Mines.

Aggregate Production at Hoover Dam

By Thomas M. Price,* Boulder City, Nev.

(New York Meeting, February, 1934)

Sand production for use in hydraulic concrete is probably the most important element affecting the physical characteristics of the resulting concrete in the opinion of most engineers, aggregate producers and contractors of today, who are engaged in the effort to produce stronger concrete per unit of cement used. Their opinion is based on both laboratory and field results. In the past, proper cement production has been the object of most of the study given this subject and while it is still studied by the extensive and well equipped laboratories that are a part of every cement mill, as well as by all governmental laboratories, at the present time the object of most of these studies has been in the field of low heat and quick setting qualities rather than the effect on the strength of the concrete.

It is recognized that concrete must have two important qualities; namely, workability with the least water possible, and uniformity. These qualities have been accepted as axiomatic and possible to obtain if the foreman in charge knew his business. Now, however, with concrete produced by plants that mechanically weigh and chart all the ingredients, including water, of a batch of concrete, the foreman cannot cut and fit his aggregates, cement and water to fit his own desires and the secret of the important qualities of workability and uniformity are found to lie largely in the sand, hence the attention given its production at the present time.

With this condition in mind the engineers of the United States Bureau of Reclamation and the Six Companies Incorporated in designing the plant for producing the aggregates for the Boulder Canyon Project included every possible means for the production of a sand of a constant and acceptable fineness modulus as well as a constant moisture content, and for attaining these results with as few detrimental ingredients, like silt and vegetable matter, as possible.

This aggregate plant (Figs. 1 and 2) is in Hemenway Wash, 7 miles north of Boulder City. It is 4 miles upstream from the low-level mixing plant. At this screening plant 8,234,000 tons of raw aggregate are being

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* Superintendent of Aggregates and Railroad, Six Companies Inc.

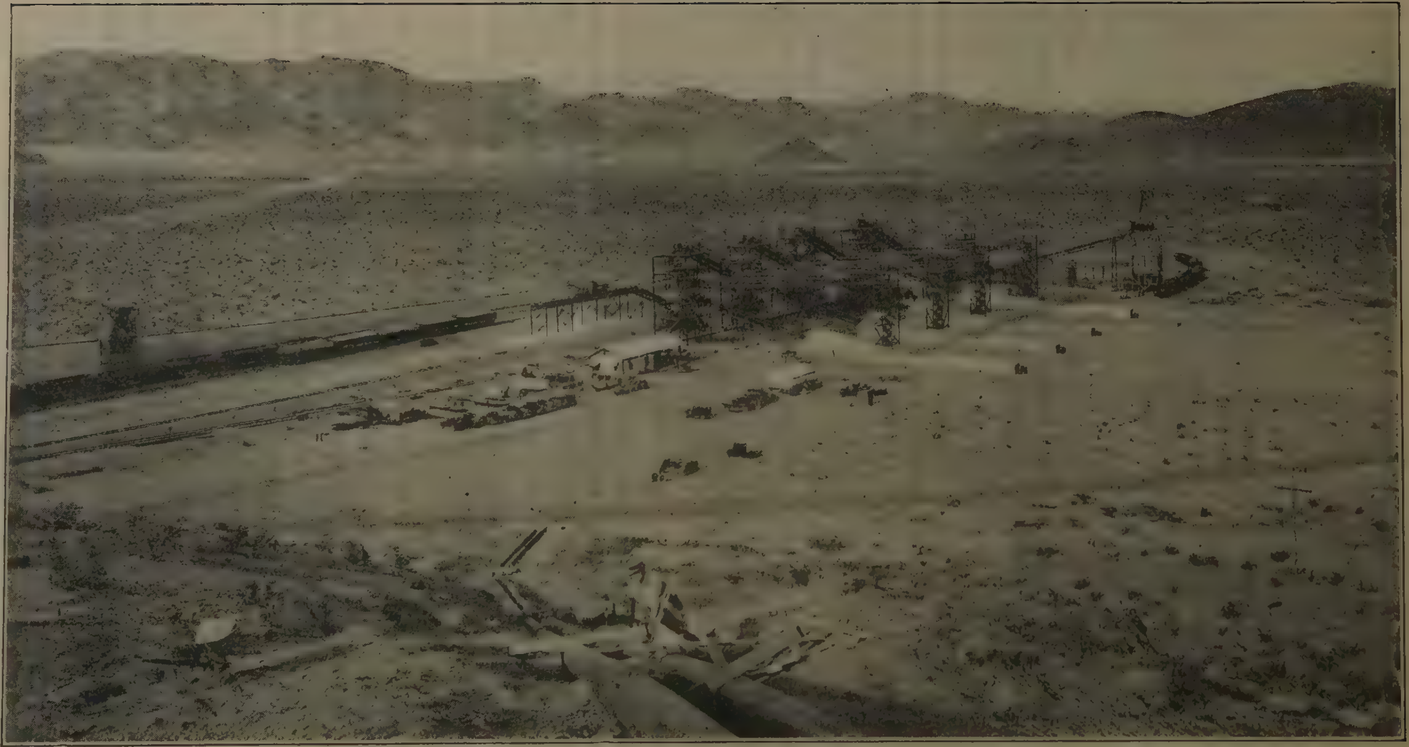

FIG. 1.—GENERAL VIEW OF AGGREGATE PLANT AT HOOVER DAM, TAKEN FROM THE DORR TRACTION THICKENER. Shows general layout of screening plant and surrounding country. The two pipes are the 16-in. fresh-water line to the plant and the 16-in. return line from all washing operations.

screened, washed and graded into four sizes of gravel and one of sand, used in the pouring of 4,400,000 cu. yd. of concrete for the Hoover Dam and its appurtenant works.

The plant was designed for an hourly production of 1000 tons, but equipment was installed for providing only 500 tons per hour, the additional equipment to be added if the plant could not furnish finished material fast enough to supply both the high-level and low-level mixing plants. However, this installation has averaged on several occasions 800 tons per hour, and the daily average through a monthly period is never less than 700 tons per hour. The high production to date was in March, 1934, when 476,000 tons was produced.

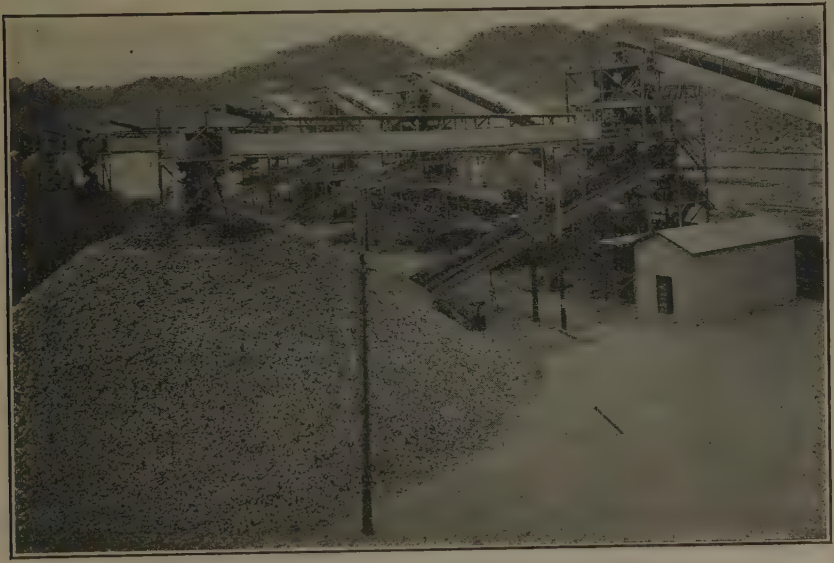

FIG. 2.—VIEW OF PLANT FROM CRUSHER BIN.

Cobble-storage pile is in foreground, tripper and storage extension having been added after this picture was taken. All four plant storage piles are shown. Building on right is main switchhouse.

GRAVEL PIT

The gravel pit is approximately 7 miles upstream from the gravel plant on the Arizona side of the Colorado River. The railroad crosses the river on an 1120-ft. wooden pile trestle. It was first proposed that a steel suspension bridge supporting a belt conveyor be constructed across the river, because of the danger of a standard-gage railroad trestle being washed out during the spring flood periods. However, it was found that the pile trestle would be the cheapest in over-all cost, even if it should wash out once a year during the three years the pit would be in operation. The possibility of its failure was one of the reasons for storage of raw aggregate near the gravel plant.

The depth of the pit deposit ranges from 10 to 35 ft., the Government requiring that the top 2 ft. of overburden be stripped and discarded because it contains an excessive amount of silt.

Material is mined by a No. 125 Marion electric dragline, equipped with a 5-yd. Esco bucket and an 80-ft. boom. A 3-yd. No. 490 Marion electric dragline is used for stripping and also acts as an alternate loading dragline in case of a breakdown of the 125 dragline.

The gravel is composed largely of hard igneous and sedimentary material and is remarkably well graded to fit the requirements of the work.

Raw material is loaded directly into Western drop-door side-dump cars of 30 cu. yd. capacity. Four Mikado-type locomotives are used for this haul from pit to plant, each hauling a train of 9 cars. Loading time per train averages 50 min., and the average production of the dragline is 650 tons per hour, or 10 cars, which is the approximate hourly capacity of the gravel plant. The railroad haul from pit to plant has an upgrade of 1.75 per cent, requiring about 30 min. each way.

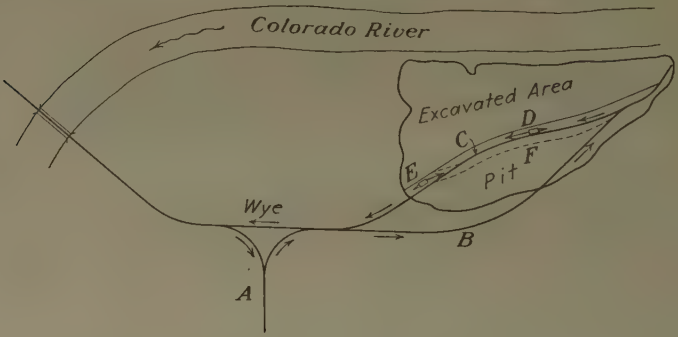

FIG. 3.—ARIZONA GRAVEL PIT, SHOWING TRACK MOVEMENTS.

As mentioned before, 1,500,000 tons of raw material from the pit has been stored near the gravel plant, to guard against a possible trestle washout and also to furnish material for plant production after June, 1934, when storage of water to elevation 950 was scheduled to start. The pit will be flooded at that time because the surface elevation of this deposit is 765 feet.

The method used in digging the pit is shown in Fig. 3. The train heads into wye *A*, backs down track *B* and heads in on track *C* behind train already being loaded when the dragline is in position shown by *D*. This system causes a minimum of delay to the shovel because one train pulls in as the loaded train pulls out. When nearing the end of the cut, approximately 500 ft. from the end, the dragline turns around and moves to *E* position and digs back the other way. In the meantime track *C* is being thrown over into position *F*. While this track is being thrown, trains back into track *C* instead of using *B* as a run-around. Incoming trains wait on wye *A* until loaded trains pull out. Moving dragline from position *D* to *E* requires only 20 min., thereby making pit production practically constant during this track-moving period. There is a 5-min. waiting period between trains while track *C* is being thrown, caused by one train waiting for the loaded train to clear the wye switch. This method of digging is made possible by the formation of the pit, which is very narrow at both ends and wide in the center.

AGGREGATE PLANT

Possibly the most outstanding feature of the plant (Fig. 4) aside from the excellence of the product is the degree to which it is mechanized. Being equipped with unit drives, the motor control of the entire plant can be centralized at one point or control tower. The plant operator is in this control tower on the top deck of the scalping-screen station, where he has full view of all operations (Fig. 5). Here are 34 push buttons, which control every motor throughout the plant except those used for loading. The entire plant is electrically interlocked, so that the operator cannot make mistakes. The plant must be started at the last conveyor and all succeeding motors started in their order, ending with the main or *A* conveyor under the track hopper. Consequently, in order to stop the entire plant, the operator has only the last motor to stop, which breaks the entire circuit. This automatically does away with all danger of piling up as a result of a motor failure, and keeps labor costs as low as possible, only a small operating crew being necessary.

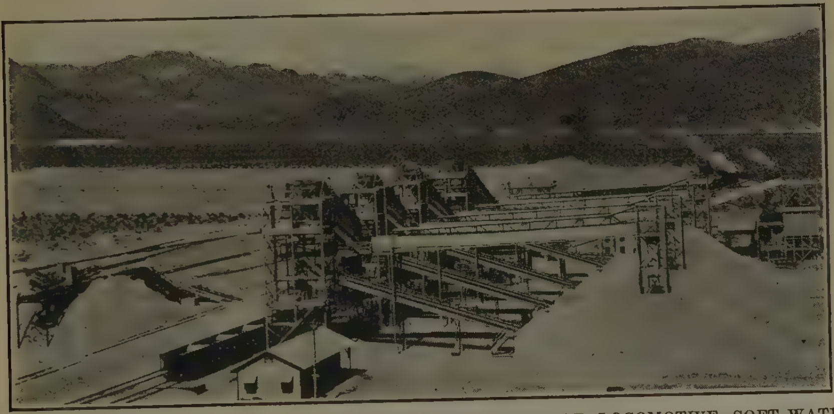

FIG. 4.—VIEW OF GRAVEL PLANT TAKEN FROM TOP OF LOCOMOTIVE SOFT-WATER STORAGE TANK.

Rock-storage piles are at right and sand storage at left. Plant office in foreground. Raw aggregate storage piles in upper center, with locomotive crane raising track. Space cleared off at left center is for future raw storage.

Raw material from the pit is dumped into a 25 by 130-ft. depressed *V*-shaped track hopper. (Fig. 6.) Side dumps or hopper cars may be dumped into the hopper, as tracks run full length on either side and there is also one across the center. A grizzly constructed of 90-lb. rail with 12-in. openings is used to keep any excessively large cobbles or conglomerate from passing into the hopper and plugging the feeder gates in the bottom. The material held on the grizzly is approximately 0.1 per cent. For this reason no steps were taken during the erection and layout of the plant to provide for its being run through the plant. The installation of an additional crusher and conveying equipment would have cost far more than the saving effected by the use of this material. The retained material is piled at one side, loaded by railroad crane into cars and used for railway bank protection.

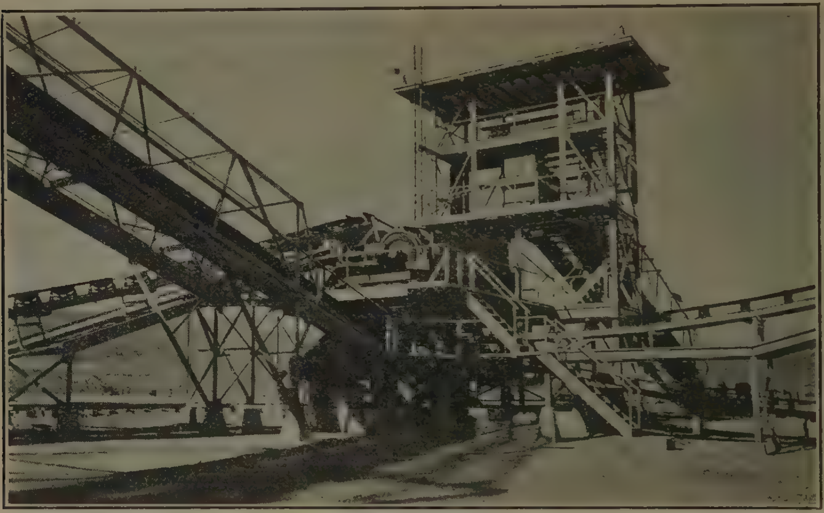

Fig. 5.—Control tower and scalping-screen station. Control station is on top deck of scalping-screen structure.

Upper right conveyor leads to crusher bin and lower right is return belt. Upper left conveyor leads to tower No. 1 and lower left is conveyor *K*, which returns all undersize from loading screens to be re-run through plant.

Material from this hopper is fed to conveyor *A*, a 42-in. eight-ply rubber conveyor belt, by means of nine Traylor vibrating pan feeders, which are operated in units of one to three at a time throughout the length of the hopper tunnel. These feeders are actuated by a direct-current magnet dampened by alternating current to control the vibration of the pan, thereby regulating the flow of raw material on to the belt. No gates are necessary because the slope of the pan is at such a degree that when the current is shut off the material ceases to flow, the angle of the straight-line opening being less than the angle of repose. Raw material is conveyed to the scalping or control tower and into a 60-in. by 16-ft. Bodinson rotary trommel screen having two 3-in. round-opening

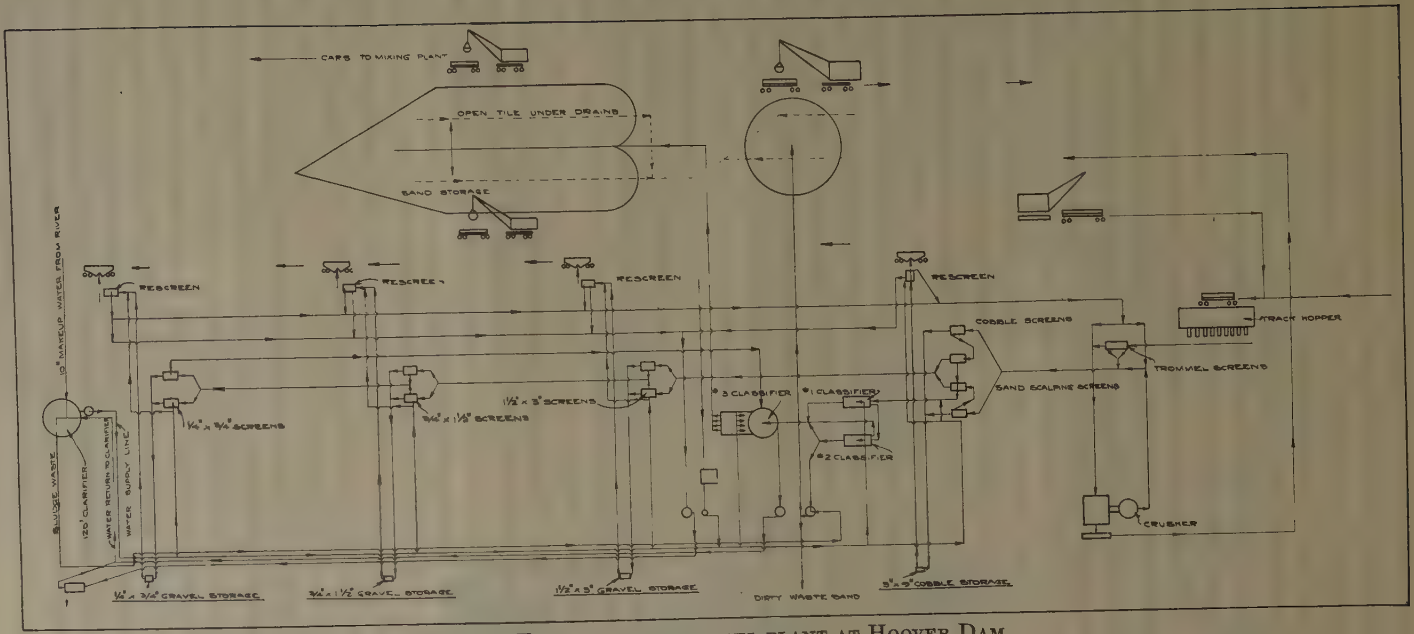

FIG. 6.—FLOW DIAGRAM, GRAVEL PLANT AT HOOVER DAM.

screen sections and two 8-in. square sections of manganese plate. The overs, or plus 8-in. square, are discharged to conveyor *B*, a 30-in. conveyor belt running at a right angle to the plant, to the cobble-storage bin, which also acts as a balancing bin from all loading screens. A wattmeter is used in the control room to show the load carried on the main or *A* belt. At light feed, approximately 300 tons per hour, the meter will show 150 watts and at full load, 750 tons per hour, will show 400 watts. The ratio of the load carried on the belt to the reading of the wattmeter is not proportionate at all times, as pit run carrying an overabundance of sand will show a much greater watt consumption than a like amount of coarser material; however, it is close enough to enable the operator to maintain constantly any set tonnage through the plant. The capacity of the plant is governed by the capacity of the sand classifiers, sandy material requiring a much lower feed. For this reason the classifiers are equipped with a loadmeter showing how near the machine is operating to capacity. If too much material is coming through the classifier, the operator signals the plant operator, who in turn cuts the feed to the main belt; if not enough material is entering the classifiers, the feed to the main belt is increased. The plant operator keeps a report of the different readings of the wattmeter during his shift, and also a report showing the number of trains from the pit, number of cars dumped into the track hopper, dumping time, delays and number of cars run through the plant during his shift. These records are kept on a comparative sheet showing how each shift operates and what their respective production is according to tons per hour produced, thus causing a friendly rivalry between shifts for greater production and least delay.

Cobbles and undersize from the crusher bin pass into a Bodinson reciprocating feeder and are fed into an Allis-Chalmers 16-in. Superior McCully gyratory crusher, set to deliver a 3-in. maximum product. Crusher material is returned by conveyor *C*, a 24-in. conveyor belt, and joins the throughs of the trommel screen. Provision was made for the installation of a secondary crusher at this point if it should be found that the pit run did not contain sufficient $\frac{3}{4}$-in. and $1\frac{1}{2}$-in. gravel to meet concrete requirements.

From the trommel screen the material is fed through a chute on to conveyor *E*, a 36-in. conveyor, and taken to No. 1 tower. Here the material passes through a divided chute. All divided chutes have a gate that can pass all material on to one screen or the other, as desired. The reason for this is that in case of a screen failure, or change of screen cloth, all material coming into any one tower may be run over either screen during the repair period, thus insuring plant operation at all times. The regulation of material passing on to the screens means that each screen carries half the load and that one screen will not be overworked more than the other. This factor insures lower maintenance costs and better screening of the material.

The chutes in tower No. 1 feed two 4 by 12-ft. double-deck Symons vibrating screens having a 3½-in. opening plate on the top deck and a 1½-in. opening on the bottom deck. Double-deck screens are used throughout the plant, as they give a more exact screening while handling a larger tonnage than single-deck screens. The top deck scalps the incoming material, removing the larger size and passing the remainder on to the smaller openings on the bottom deck, which is the actual sizing deck. All screens are suspended by springs mounted on each corner, to prevent vibration from being transmitted to the towers. The overs from these two screens pass to conveyor *F*, a 30-in. conveyor belt running horizontally at right angles to the plant and parallel with the crusher conveyor. This conveyor carries the material 125 ft. to a rock ladder (Fig. 7), from which it is distributed by a Link-Belt tripper to the plant storage pile, which is approximately 125 ft. long and has a capacity of 15,000 live tons. A tripper is used on the cobble-storage pile, as it was found less expensive to reclaim by this method than by stocking excess production by railroad cars. Cobbles, because of their large size, are difficult to rehandle by clamshell; also, breakage is far higher when stocking from railroad cars, because of the fall when dumped. Another important reason for increased stocking of this material at the plant is the variation of the pit. The north end yields an excess of cobbles and the lower, or south, end is short of this size. Increased live stocking capacity tends to even out this variation.

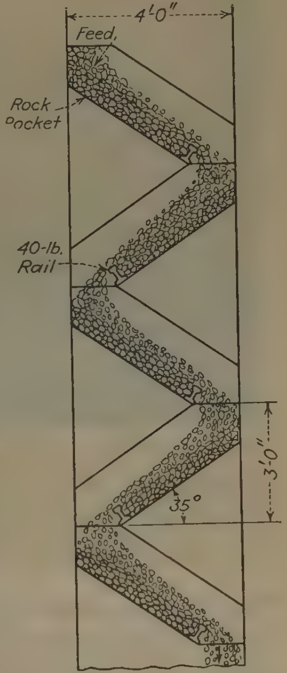

Fig. 7.—Action of rock ladder.

The rock ladder is a series of cascades (Fig. 7), the material falling from one step to the next and so on down, in order to eliminate to as great an extent as possible the breakage caused by the fall into the stock-piles. At the lower end of each step is a piece of 40-lb. rail, which backs up sufficient material so that the aggregate falls on itself rather than on the steps, causing less breakage and also lowering the cost of replacements for new liners in the ladders. By the use of the rock ladder principal breakage resulting from the fall into the storage piles runs from 3 to 5 per cent, which is very low. Keeping breakage at a minimum saves the cost of rerunning undersize through the plant. Material must not contain over 10 per cent undersize when loaded, as ruled by the Bureau of Reclamation.

The throughs from these two screens in tower No. 1 are chuted to the deck below, where the material is run over two Symons screens, each 4 by 12 ft. in size. The scalper deck of each screen contains 1-in. openings and the bottom deck has a ¼-in. sand screen. The overs from these screens are taken by conveyor *H* on a 30-in. conveyor belt to tower No. 2. The throughs, being one-quarter minus product, are washed down a 16-in. steel launder to Dorr sand classifiers, the operation of which will be described later.

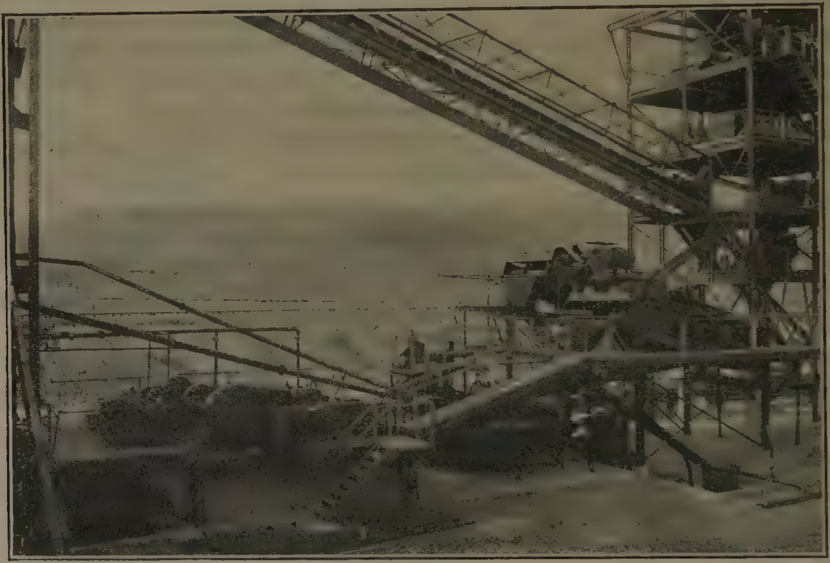

FIG. 8.—DORR SAND CLASSIFIERS. LAUNDER LEADING FROM SAND SCREENS IN TOWER NO. 1 IS FEEDING INTO NO. 1 CLASSIFIER AT RIGHT. NO. 2 CLASSIFIER IS NEXT TO NO. 1.

Waste sand launder leading to tank in pump pit at lower right. Material from one or two classifiers flows into bowl in center, to be raked out as finished product by rakes at left. Return pipes from re-screens are shown in upper left.

The 3 by ¼-in. material arriving at tower No. 2 is again divided and run over two double-deck Symons screens having 1½-in. openings on the top decks and ¾-in. openings on the bottom. The overs from both decks are chuted to conveyor *F*-1 and carried on a 30-in. conveyor belt and stored by means of a rock ladder into a conical pile of 9000 live tons capacity. A tripper is used only on the large material, 3 by 8 in., for reasons outlined above.

The throughs, or 1½-in. minus from the screens in tower No. 2, are transferred to conveyor *I* and taken on a 30-in. conveyor belt to tower No. 3. Here the material is run over two Symons double-deck vibrating screens, the top deck having 1½-in. openings and the bottom deck ¾-in. The overs, or 1½ by ¾ in., are carried by conveyor *F*-2 and stored in conical pile similar to that described before.

The throughs, or ¾-in. minus, are carried by conveyor *J* on a 30-in. conveyor belt to tower No. 4. Here the material passes over two Symons screens having ¼-in. top decks and ⅛-in. bottom decks. The overs are carried by conveyor *F*-3 to a conical storage pile and stored as before. The ¾ to ¼-in. material is wet screened to wash off any remaining sand which is carried with the water through a 6-in. steel pipe back to the sand classifiers between towers 1 and 2.

The plant is equipped with four Dorr classifiers (Figs. 8 and 9) designated by numbers 1, 2, 3 and 4. Classifiers 1 and 2 are of the 8-ft. double-rake type, 3 and 4 being of one unit consisting of a 20-ft. bowl and a 16 by 33-ft. rake classifier. These classifiers consist of a set of mechanically operated reciprocating rakes suitably installed in a rectangular inclining settling box. A proper amount of water to produce the necessary slime consistency is applied immediately after screening, and this water is sufficient to launder the sand into No. 1 classifier, where it is distributed across the width of the tank near the overflow end. The heavier quick-settling material settles to the bottom of the tank, to be moved up the slope and discharged into a chute leading to the bowl. The rake motion agitates the material in the box, throwing the lighter or finer material into suspension to be carried off into the overflow at the lower end of the tank that discharges into No. 2 classifier. The sand product of No. 1 classifier is ¼ in. to No. 28 standard mesh. No. 2 classifier is operated on the same principal as No. 1, producing a sand product of 28 to 48 mesh, which is carried by rakes, moved up the slope and discharged into a chute leading to the bowl. The slimes, or minus 48-mesh product kept in suspension by the agitation of the rakes, overflow at the back end and run down a launder where they

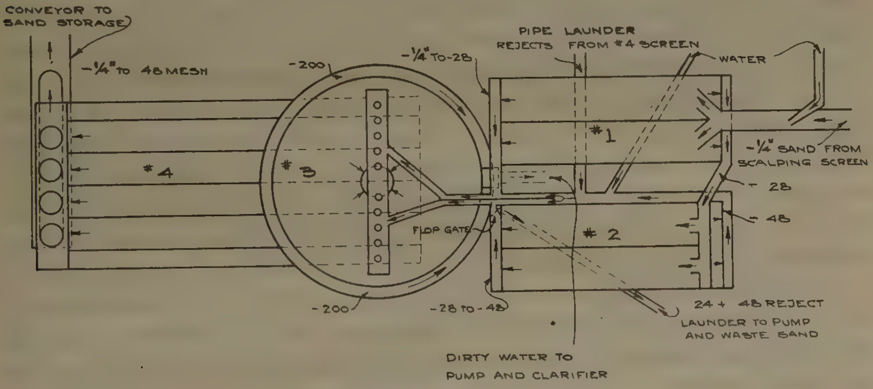

FIG. 9.—SAND CLASSIFIERS.

join the finished product from the front of classifiers 1 and 2, and from here the combined material passes into a feed box over classifier 3, or the bowl. The feed box is installed at right angles to the incoming chute, and extends across the diameter of the tank, to distribute the material evenly across the bowl. If incoming sand were not spread out the coarser, heavier material would all settle in the center and the fines would carry to the outside, thus making an uneven gradation of the finished product at the front of No. 4 classifier.

Classifier No. 3 is a two-stage baffled return classifier and consists of a comparatively shallow tank with revolving rakes, which convey the material to a central opening in the bowl bottom where it passes into No. 4 classifier below. The bowl classifier has an overflow launder to carry away the dirty water and the minus 200-mesh product, which is wasted. No. 4 classifier has reciprocating rakes, as have Nos. 1 and 2, which carry the finished sand up an incline to be discharged as a drained, finished material.

Water plays a very important part in the making of the finished sand, all classifiers being well equipped with sprays under high pressure. The water is cleaned by the water plant, which will be described later. Bureau of Reclamation specifications for sand are as follows: "It must be free from injurious amounts of dust, lumps, soft or flaky particles, shale, alkali, organic matter, loam, mica, or other deleterious substances. The sand for concrete shall have a fineness modulus of not less than 2.65 nor more than 2.85. The fineness modulus is determined by dividing by 100 the sum of the percentages retained on Tyler standard sieves numbers 4, 8, 14, 28, 48 and 100."

Sand modulus, which must approximate 2.70 by these specifications, may be controlled at the front of classifiers 1 or 2 by wasting part, or all, of the material produced, by means of flop gates in the chutes leading to No. 3 classifier.

All water entering the classifiers comes from the launder from the sand screens, the return pipe from all gravel washing operations, or the sprays on the front of No. 4 classifier. Water carrying the sand from the sand screens is regulated by remote control from the classifiers. The more water added at this point, the more material will be carried out through No. 1 classifier. This tends to make the whole machine flexible in its operation, necessitated by changes in the incoming sand. By keeping the water low in the case of too much fine sand, the material will be held back in classifier 1 and carried over into classifier 2, where it may be wasted. If incoming sand is good as to grading, the control of the water tends to even the distribution into the two classifiers, making them both carry the same or equal load.

Water entering No. 4 classifier through high-pressure sprays must be watched carefully to avoid plugging the machine. These sprays are

TABLE 1.—*Typical Hourly Test*

MODULUS OF RAW SAND AS IT COMES FROM GRAVEL PIT

	Per Cent	Cumulative Per Cent
Pass ¼ in. retained on No. 4	5	5
Pass No. 4 retained on No. 8	11	16
Pass No. 8 retained on No. 14	7	23
Pass No. 14 retained on No. 28	7	30
Pass No. 28 retained on No. 48	40	70
Pass No. 48 retained on No. 100	21	91
Pass No. 100 retained on No. 200	6	
Pan	3	
Total	100	
Modulus		2.35

This material is fed to No. 1 classifier, making roughly a separation in the underflow around the 48 mesh. Note that sand modulus above is too low to meet specification of 2.70.

MODULUS OF UNDERFLOW

	Per Cent	Cumulative Per Cent
Pass ¼ in. retained on No. 4	6	6
Pass No. 4 retained on No. 8	16	22
Pass No. 8 retained on No. 14	10	32
Pass No. 14 retained on No. 28	12	44
Pass No. 28 retained on No. 48	39	83
Pass No. 48 retained on No. 100	13	96
Pass No. 100 retained on No. 200	3	
Pan	1	
Total	100	
Modulus		2.83

Modulus of this sand is too high to meet specification of 2.70. To correct this the overflow of the No. 1 classifier is fed to No. 2 classifier.

RESULTING MODULUS OF UNDERFLOW OF CLASSIFIER No. 2

	Per Cent	Cumulative Per Cent
Pass ¼ in. retained on No. 4	0	0
Pass No. 4 retained on No. 8	0	0
Pass No. 8 retained on No. 14	2	2
Pass No. 14 retained on No. 28	11	13
Pass No. 28 retained on No. 48	56	69
Pass No. 48 retained on No. 100	23	92
Pass No. 100 retained on No. 200	6	
Pan	2	
Total	100	
Modulus		1.76

TABLE 1.—(*Continued*)

In order to bring modulus of underflow of No. 1 classifier to specification requirements, it is necessary to add enough of underflow of No. 2 classifier to underflow of No. 1 classifier to bring modulus down to 2.70, and waste remainder. This equals in amount 30 per cent of the No. 1 underflow, estimated as follows:

	(1) Feed Units from No. 1	(2) Feed Units from No. 2	(3) 30 Per Cent No. 2	Add Columns (1) and (3)	Per Cent	Cumulative Per Cent
Pass ¼ in. retained on No. 4	6	0	0	6	5	5
Pass No. 4 retained on No. 8	16	0	0	16	13	18
Pass No. 8 retained on No. 14	10	2	0	10	8	26
Pass No. 14 retained on No. 28	12	11	3	15	13	39
Pass No. 28 retained on No. 48	39	56	17	56	45	84
Pass No. 48 retained on No. 100	13	23	7	20	16	100
Total	92	92	27	123	100	
Resulting modulus						2.72

This modulus is close enough to comply with specifications. Screen test of resulting sand is as follows:

	Per Cent	Cumulative Per Cent
Pass ¼ in. retained on No. 4	5	5
Pass No. 4 retained on No. 8	13	18
Pass No. 8 retained on No. 14	8	26
Pass No. 14 retained on No. 28	13	39
Pass No. 28 retained on No. 48	44	83
Pass No. 48 retained on No. 100	15	98
Pass No. 100 retained on No. 200	2	
Pan	0	
Total	100	
Modulus		2.69

Showing that final result is close to estimated result and that finished sand complies with specifications. Note that classifiers 3 and 4 have removed small percentages of light fines and brought the 100-200 to less than 5 per cent, which is specification limit, as well as washed the sand so that the pan is negligible.

18 in. from the discharge end of the classifier and are there for the purpose of a final washing of the finished product. Enough water must be applied to perform this washing thoroughly, but if too much water is applied the sand has a tendency to back up, piling over the reciprocating rakes at the lower end and plugging the machine. It can be readily seen that the classifiers are the heart of the entire plant operation.

A typical hourly test made to determine necessary adjustments in order to obtain satisfactory sand product is shown in Table 1.

The finished sand is taken by 30-in. conveyor *M* through a tunnel underneath the yard tracks, running horizontally at a right angle to the south of the plant and transferred to conveyor *N*. This conveyor, running parallel to the plant, transfers the sand to a Bodinson airplane type tripper (Fig. 10) moving continuously back and forth along a 200-ft. sand-storage pit. The sand is discharged at right angles both to the right and the left by conveyors *O* and *O*-1. Sand is stored by this method rather than binned because of the selective feature of clamshell recovery. Sand is stored in three separate divisions of the tripper track, one section for each shift. Sand loaded for shipment to the mixers is taken from the pile that has drained until moisture content is acceptable, it being apparent that a very even moisture content of the outgoing sand may be obtained, which is very important from the standpoint of uniformity. The reason for the continuous back and forth movement of this tripper is to thoroughly mix together any temporary variation in the sand product, thus keeping the fineness modulus of the total storage uniform.

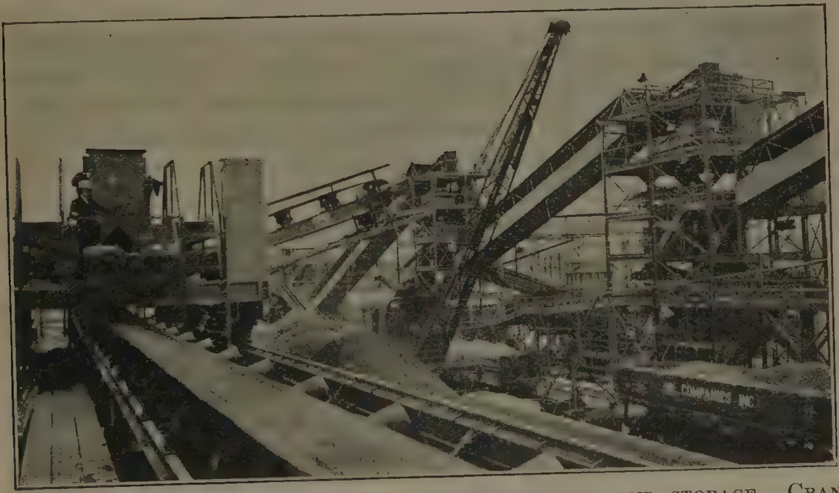

FIG. 10.—AIRPLANE-TYPE BODINSON TRIPPER USED FOR SAND STORAGE. CRANE IS LOADING SAND CARS FOR SHIPMENT.

Sand is reclaimed from storage by a 30-ton Brownhoist locomotive crane using a six-cylinder Atlas Imperial diesel engine for power. An Owens clamshell (1½-yd.) is used for this purpose. The locomotive crane spots its own cars while loading, thus eliminating an extra yard engine. The sand-storage pit (Fig. 11) has a concrete base containing drain tile to carry away all drainage from the wet sand.

Finished coarse aggregate is reclaimed for shipment from the plant stockpiles by 30-in. conveyors *G*, *G*-1, *G*-2 and *G*-3, which run from tunnels under the plant storage piles Nos. 1, 2, 3 and 4 respectively. The material is conveyed to the second deck of each tower, where it passes over a double-deck Symons vibrating screen. All undersize from the loading screens is taken by 30-in. conveyor *K* back to the scalping screen and again run through the plant. The overs from these screens in towers 2, 3 and 4 are loaded by 48-in. conveyors *L*-1, *L*-2 and *L*-3 into railway cars either for storage or mixing-plant use.

The material loaded from No. 1 tower, 3 to 9-in. rock, is divided and run over two vibrating loading screens of Robins manufacture. The material from one screen dumps directly into cars next to the tower. Material from the other screen passes on to a swing conveyor and from here may be loaded into the same car or cars spotted on the second loading track. Shuttle conveyors in towers 2, 3 and 4 are reversible and can load material into cars on either side of the plant. Material on all loading belts is subjected to wash water at a high pressure to remove any sand or silt remaining. Loading belts are perforated over their entire surface by ¼-in. holes spaced 1 in. apart, for drainage of the water from the washed product. Material is washed here rather than as it passes over the re-screens because experience has shown that wash water carrying accumulated dirt with it will pass off considerably faster when at rest on a flat surface than when it is running over a screen. A large saving of water is effected by this method also, as running material has a tendency to carry wash water along with it.

There are two stub track classification yards having four tracks each (one for each size of gravel) where the loading engine spots loaded cars to await shipment to either storage piles or the mixing plants by aggregate trains. In general the yard movement is comparable to the gravel plant in that cars of raw material enter one end and cars of finished material are released from the opposite end.

The plant operating crew consists of 16 men on each shift, as follows: Foreman, plant operator, classifier operator, electrician, screen tender, hopper tender, Brown-hoist operator, Brown-hoist oiler, top oiler, bottom oiler, loader, loader's helper, four clean-up men. The plant runs 24 hr. per day—three shifts of 8 hr. each. One-half hour is taken for meals on each shift. So that no time will be lost during this period, the lunch time is staggered, the operators of the more important parts of the plant having men broken in to their jobs. This plan also assures no loss of time due to sickness or any other disability.

Westinghouse electrical equipment is used, all motors being of the totally enclosed, fan-cooled, ball-bearing type ranging from 5 to 250 hp. The total connected load at the gravel plant is 1325 hp. and there is approximately 55,000 ft. of electric conduit in use. The main switch-house is of steel construction with metal lath and plaster for the walls. The only ventilation is through a ventilator in the roof and a cooling system. The building is kept as airtight as possible to keep out all dust, which causes contacts to pit and burn. The cooling system was installed to prevent the overload relays from becoming hot enough to kick out during the extreme heat of the summer months, the temperature in the sun sometimes being as high as 140°.

The main switchboard is 8 ft. high by 20 ft. long and is so wired and constructed that any motor in the plant can be controlled from this point, the control tower, or at the motors themselves. The switch-house also contains the motor-generator sets that operate the vibrating feeders in the track hopper.

A 75-hp. stationary boiler is in use during the winter months, furnishing steam at all idlers and head pulleys to keep the water on them from freezing and thus causing the belts to slip. Winter temperature approximates 10° minimum.

All conveyor belting was furnished by the Pioneer Rubber Co. of San Francisco. Belting ranges from four to eight ply and there is approximately 6500 ft. in use. All conveyor pulleys and take-ups were furnished by the Link-Belt Co. and Rex Stearns idlers were furnished by the Chain Belt Co. Falk speed reducers and high-speed couplings were used.

The plant is constructed of reinforced concrete and structural steel with timber floors, railing and steps. Although it is a temporary plant, this construction was used to eliminate fire hazard and to preserve alignment.

WATER SYSTEM

The unusual characteristics of the Colorado River made necessary the installation of the finest and most efficient types of equipment for clarifying and purifying the water. Even these, however, fail during periods in the early fall when the river turbidity is over 150,000 parts per million, due to cloudbursts. Then the silt stays in suspension and will not settle, even with the use of chemicals, fast enough to furnish clear water for both the gravel plant and the railroad. During these periods water is hauled from Las Vegas, Nev., for use in locomotives and for drinking.

The pumping system consists of one pump at the river and two booster stations approximately ¾ mile apart. Each pump station contains three Byron-Jackson *S*-4 centrifugal pumps with a lift of 75 lb. and a capacity of 450 gal. per min. per pump. Two pumps are in use at all times and the third is for spare during repair periods.

Special Byron-Jackson sand pumps having Nitraloy runners are used. This is necessary because of the high sand and silt content of the river water. Runners in the pumps last from six to eight weeks. When runners are replaced it is also necessary to build up the case, which is done with bronze, which necessitates the pre-heating of the whole case to eliminate internal strains.

The pump station at the river is mounted in a pumphouse, which in turn is mounted on a 31° inclined set of rails, so that it may be raised and lowered at will to take care of the rise and fall of the river during flood periods. A floating weir is used at the intake so as to keep the top 5 or 6 in. of water available for the water supply. This is done to make clarification of the river water faster. At the river's lowest point it carries about three per cent of silt.

The water is pumped from the river station through a 12-in. O.D. pipe line, through the two booster stations to the clarifier, approximately 2¼ miles uphill adjacent to the gravel plant, the total rise from the river being 485 ft. Clarification of the river water is accomplished by a Dorr traction thickener (Fig. 12). Water enters the clarifier at the center through a concrete pillar, which is also the center bearing for the traveling rakes. There are four rakes on the traveling mechanism. Three are of the cantilever trussed type, extending approximately one-half the radius of the tank bottom; one rake extends the full radius and has the driving mechanism on its outer end. The water entering at the center of the tank carries a large amount of sand, which settles quickly, thus necessitating the use of more rakes in the center. The actual construction of the entire mechanism can be seen in Fig. 12, which also shows clearly the location of the clarifier with respect to the gravel plant itself.

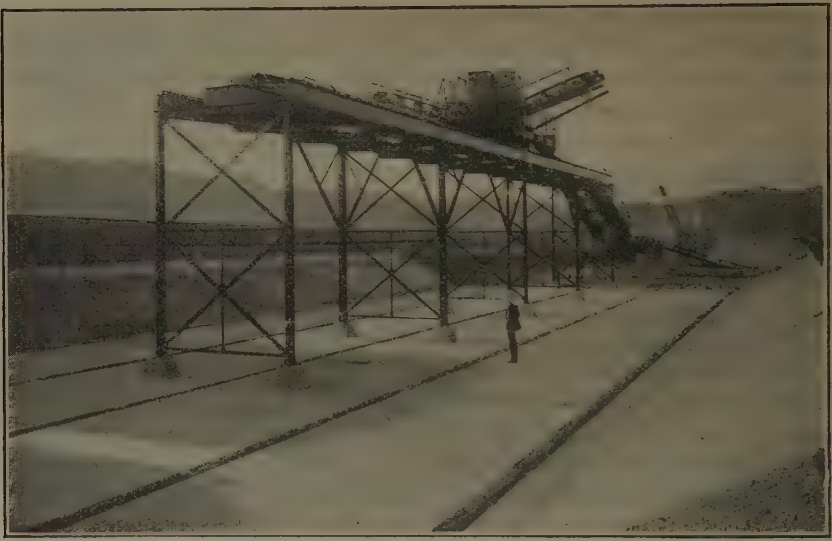

FIG. 11.—SAND-STORAGE PIT BEFORE STORAGE WAS BEGUN.

Bodinson tripper is shown on superstructure. Water-drainage system may be seen in concrete base; 90-lb. rail was concreted on all edges of drainage channels to prevent chipping when struck by clamshell while loading into cars.

The clarifier tank is 120 ft. in diameter, 12 ft. deep at the outside edge and sloping to 20 ft. at the center. The moving blades or rakes are set at an overlapping angle on steel framework, which has as its axis the intake pillar in the center of the tank. The outer end has a traveling carriage, which rotates around the tank on a steel rail mounted on the top edge of the wall. The carriage rotates around the tank at a speed of approximately four revolutions per hour.

Precipitated matter, or sludge, is carried out through two 3-in. drains set in the lowest point at the center of the tank. Two means of egress are provided: gravity flow and a Dorr twin 4-in. diaphragm sludge pump. Clarified water is discharged into a weir at the top inside edge of the tank. This weir empties into a well from which the water is distributed for plant use and for softening for locomotive-boiler use.

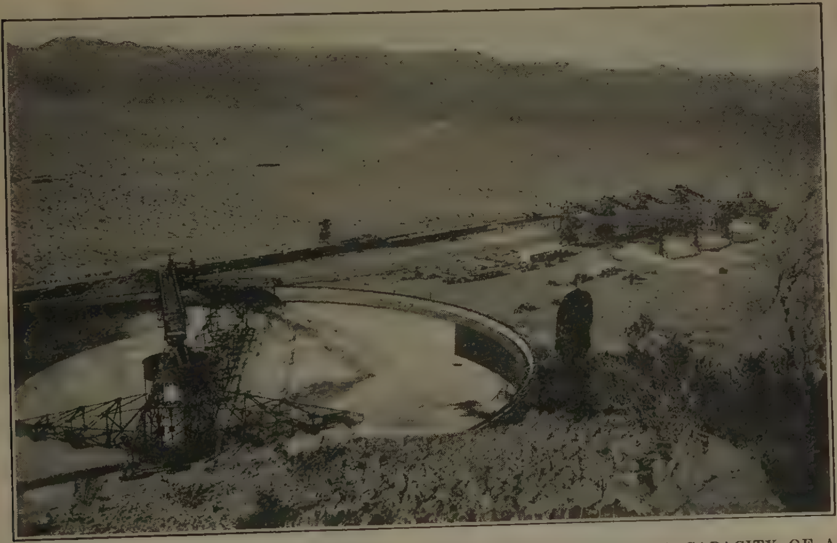

FIG. 12.—DORR TRACTION THICKENER IN FOREGROUND, HAVING A CAPACITY OF APPROXIMATELY 1,000,000 GALLONS.

Sludge is raked into center by revolving rakes and wasted. Clear water runs over into launder around top edge of tank. Location of clarifier with respect to gravel plant shown clearly. Track in upper center leads to Boulder City. Finished aggregate-storage piles above high-water line, and high-level mixing plant.

Water for plant use is carried by a 16-in. line from the clarifier. Approximately six million gallons are clarified for screening plant use every 24 hr., of which five million gallons are recovered and returned for reclarification. All sand-classifier water is returned to the clarifier through a 16-in. return line by a 10-in. Cameron centrifugal pump. Wash water from loading screens and water reclaimed from stockpiles are combined and returned to a Bodinson double-rake classifier, where the sand is removed and the water returned to the clarifier. This sand is not stored for shipment because it contains a considerable amount of scale. Plant stockpiles are equipped with sprays which keep the material wet at all times, thus removing much of the scale that is present on most of the pit-run material. The Government requires that this material be wasted, and owing to the expense of pumping water from the river, the water is removed and returned to the clarifier and the sand product is wasted. The sand is removed before the water is returned to the clarifier, as the material is too heavy to be wasted by the horizontal drainage or sludge pipes. The sand thus accumulated is carried away by the sludge water from the clarifier.

Water for railroad use and for drinking is taken from the clarifier well and run through two J-30 International filters. The water is then treated with soda ash, lime and sodium aluminate, to remove hardness and impurities. The soft water is stored in a 125,000-gal. bolted steel tank, which is connected by a 10-in. line to a locomotive standpipe. This treated water is also used for boiler washing at the roundhouse.

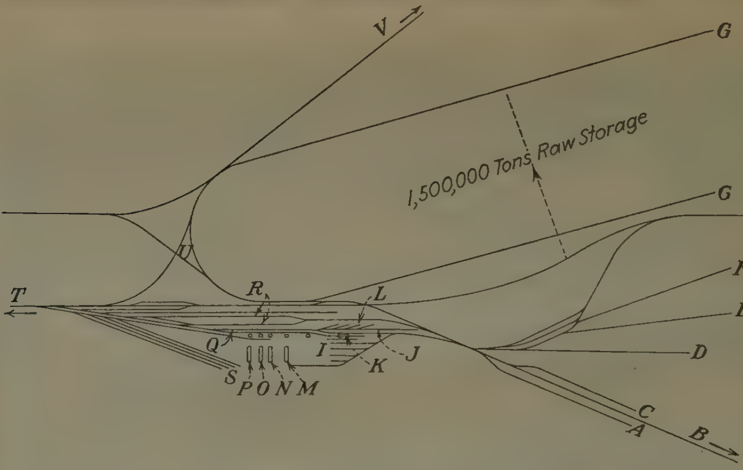

FIG. 13.—TRACK LAYOUT, GRAVEL PLANT AT HOOVER DAM.

A. Finished sand storage.
B. Pit line.
C. Finished storage No. 1.
D. Finished storage No. 2.
E. Finished storage No. 3.
F. Finished storage No. 4.
G. Raw storage, unloading.
H. Raw storage, reclaiming.
I. Roundhouse, rip tracks.
J. Hopper tracks.
K. Hopper.
L. Sand No. 1 loaded car yard.
M. No. 1 screening tower and stockpile.
N. No. 2 screening tower and stockpile.
O. No. 3 screening tower and stockpile.
P. No. 4 screening tower and stockpile.
Q. Tower loading track.
R. Sand loading track.
S. Loaded car yard, Nos. 2, 3, 4.
T. Line to low mix.
U. Wye.
V. Line to high mix.

RAILROAD

The railroad (Fig. 13) used for the transportation of all aggregates and cement and other miscellaneous items will handle 150,000,000 live ton-miles during the life of the job. The system includes some 40 miles of standard gage 90-lb. trackage. It is operated in the same way as standard railroad lines, having a dispatcher on three 8-hr. shifts, trainmaster and yardmasters. A regular train order schedule is maintained over the entire trackage and only the most competent trainmen and enginemen are employed, because of the hazardous grades, which in some cases are 3.5 per cent.

Trains are dispatched by telephone and written train orders, each station and siding being equipped with a dispatcher's telephone. Practically all lines are single track with sufficient sidings to prevent train delays. Motive power consists of five Mikado-type locomotives, ten Consolidation type and one Shay geared type.

The roundhouse, which is at the gravel plant, consists of a competent crew of boilermakers, air-brake men, machinists and mechanics, who take care of all minor repairs to locomotives. A rip track is also maintained here for the repair of railroad cars. All major locomotive repairs are done by the Union Pacific system in its roundhouse at Las Vegas, Nev., some 30 miles away. All engines are kept up to the Interstate Commerce Commission requirements and are subjected to monthly and annual inspections as laid down by the Commission.

Railroad cars used for the purpose of aggregate shipment consist of 50 hopper cars, 66 side dumps and 42 Hart selective type hopper cars.

SUMMARY

In summarizing the high lights of the construction and operation of this plant, it should be remembered that it is a project or job plant and is not to be confused with a commercial plant. While many of the schemes and principles used can and are applied in commercial plants, there are also many that would not be practicable in such plants. This plant has only one customer to satisfy, a commercial plant has many, and each customer has his own opinion of what he wants in the way of aggregates. Probably the greatest difference between the two types is that this plant has no blending facilities, because none are needed. Also, only five sizes of aggregates are made; most commercial plants have three times as many. The high lights of this production are:

1. Uniformity of product as to grading and moisture content.
2. High degree to which plant is mechanized, particularly the control features of the various units and processes.
3. The small amount of waste, both of aggregates and water, and the almost automatic means for recovering these wastes.

Vermiculite—Production and Marketing by the Zonolite Company

By William S. Steele,* New York, N. Y.

(New York Meeting, February, 1934)

Vermiculite is a nonmetallic mineral of the mica family, probably related to biotite mica. Its occurrence has been noted at numerous places in the United States and in some foreign countries, but it can be said that vermiculite is of only moderately wide distribution. A laboratory and field survey of over 60 deposits in the United States by F. E. Schundler & Co., of Joliet, Ill., indicates that the purest deposit represented in this survey is on the property of the Zonolite Company at Libby, Montana.

The Zonolite Company is commercially active and the vermiculite it produces is sold as Zonolite. Hereafter, in this report, the word "Zonolite" will be used to designate the vermiculite produced by the Zonolite Company. Values given herein to Zonolite are peculiar to the vermiculite produced from the Zonolite Company's deposits. References to "vermiculite," are to be interpreted as covering the broad field covered by the Schundler survey.

Composition and Characteristics of Vermiculite

Vermiculite is an aluminum-magnesium silicate. The following typical analysis is published by the United States Bureau of Standards: silica, 41 per cent; iron oxide, 7; aluminum oxide, 18; magnesium oxide, 21; calcium oxide, 1; alkalis (sodium and potassium), 1; moisture, 11.

Vermiculite breaks down into platelike formation in crushing and grinding operations. Its cleavage is basal. Crude, screened, dry Zonolite weighs between 80 and 90 lb. per cubic foot; its specific gravity is 2.8 and its specific heat is approximately 2. The fusion point of Zonolite is 2462° F.

Crude vermiculite contains between 7 and 9 per cent of chemically combined moisture. The instantaneous introduction of crude Zonolite into an expansion furnace at a temperature of between 1800° and 2000° F. transforms the combined moisture into steam, which in an effort to liberate itself forces an explosive expansion of the platelike crude material

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* General Sales Manager, F. E. Schundler & Co., Inc.

into a honeycombed granule not unlike an accordion and extremely brittle or friable. Immediate cooling of the expanded material toughens the granules into a durable, rubbery aggregate, ready for use.

PRODUCTION AND PREPARATION OF ZONOLITE

The production of vermiculite from the deposits of the Zonolite Company entails no mining problems. The mine is 14,000 ft. in a direct line from the town of Libby, Mont., across the Kootenai River. By road this distance is approximately 7 miles. The entire Zonolite property has been prospected. Borings have been made to a depth of 800 ft. Shafts have been sunk and numerous tunnels have been dug on this property during the past 10 years. Various geological surveys have estimated that the Zonolite property contains from 25 to 100 million tons of almost pure vermiculite.

Occasional very thin ribbons of feldspar and quartz run through the solid vermiculite face, but these occurrences are so scattered and so minute that treatment for separation of the crude vermiculite is almost unnecessary. There are wide outcroppings of vermiculite and there is seldom more than 3 ft. of overburden.

Zonolite is quarried from an open cut at the top of the mountain. The ore is easily excavated with pick and shovel, consequently very little powder is used. Operations at present are being confined to a face approximately 50 ft. wide and 200 ft. deep. Zonolite is broken down from the face, loaded into small broad-tired mine carts and hauled to the primary screens. The fines are removed and dumped, while the larger pieces are loaded into trucks for delivery to the mills, a distance of about 7 miles. The primary screening equipment is at the workings and the mills are on the railroad at Libby.

An aerial tramway, to be operated by gravity, from the pit to the railroad, directly across the river, has been planned and the right of way partly cleared for construction at an early date. Operation of this tramway, when completed, will remove the necessity of screening at the pit and will permit year-round working regardless of weather conditions. The output of predried, prescreened Zonolite will then be limited only by the capacity of the screening and drying equipment at Libby, and at a greatly reduced production cost. The present long truck haul has been a serious handicap, particularly during the winter months, for although the county roads are kept open the year round, snow banks on the private roads render them impassable at times.

The remarkable properties of vermiculite, which have rapidly attracted the attention of the insulating and acoustical industries and caused its adoption in these fields, are of interest to every engineer, regardless of the specialized field in which he operates.

EXPANSION OF ZONOLITE

The percentage of free moisture in crude Zonolite is variable, depending largely upon weather conditions. Inasmuch as free moisture is an expensive factor, Zonolite is dried before grinding and screening. The unsized crude material is passed through a rotary kiln heated by an oil-fired Dutch oven, to a temperature not exceeding 180° F. Care must be exercised not to bring the drying heat high enough to disturb the combined-moisture content and thereby retard its full expansion, yet at the same time to carry it sufficiently high to reduce the free moisture to a maximum of 3 per cent.

The expansion of Zonolite is much more rapid and complete where the input of crude material is uniformly graded. For this reason, after drying all Zonolite is passed through a hammer mill and then separated into commercial grades. The largest size of graded material passes a ½-in. mesh screen and is retained on a ¼-in. mesh screen. The intermediate grade passes the ¼-in. mesh and is retained on a No. 10 screen. Material through the No. 10 and caught on a No. 28 screen is known as "plaster size" because of its wide use as an aggregate in cement and plaster mixes. All material screened through the 28-mesh screen is handled and expanded as one grade known as "fine" or "small" grade.

In addition to these four standard commercial grades, several uses have been developed for both crude and expanded pulverized Zonolite and for materials screened to other definite specifications. These special gradings are not carried in stock but are processed to meet specifications as ordered.

Lightness in weight, toughness and complete expansion are the first consideration in the expanded material and every step in the Zonolite process is planned with these ends in view. A considerable separation of foreign rock and particles of overburden is accomplished in the grinding and screening procedure. Overburden and organic matter that survives the grinding and screening operations is ultimately consumed in the expanding furnace or dropped out in the suction conveyors.

For years, the expansion of Zonolite was performed in a rotary kiln, maintained at a temperature of 800° F. The crude ungraded material was introduced at the cold end of the kiln, traveling and tumbling slowly through the heat. The resultant product was heavy. It was only partly expanded and it suffered considerable compression in its journey through the kiln. F. E. Schundler, of Joliet, Ill., and his chemical director, P. S. Denning, in 1924 conceived the theory that maximum expansion and freedom from compression were possible only when the ore was accurately graded, instantly passed into a soaking heat above 1600° F., taken out quickly and immediately cooled.

The Schundler Expander

The first pilot plant constructed on this theory was a direct-fired gas oven constructed over a rapidly vibrating jigger base. This unit proved the correctness of the Schundler theories, but developed several impractical angles, which had to be overcome by later improvements.

Next, an electrical expander was built, also with a vibrating table or jigger base. This machine proved to be quite satisfactory in so far as the finished product was concerned, but impractical from the standpoint of power cost. Where electricity can be obtained at a cost of less than ½ cent per kilowatt hour, this unit is an excellent one.

Then followed a long series of experimental units combining various theories of heat treatment with numerous types of construction. Each development corrected faults of previous equipment but in turn developed disadvantages of its own.

For the past two years, the Schundler company has standardized on a direct-fired inclined gas furnace, which has been very satisfactory. A measured feed of crude Zonolite is carried by bucket conveyor to a small hopper directly over the furnace. The hopper discharges the same measured feed into the top of the furnace, where it is met by a battery of six gas burners firing directly against the ore input. The material travels down the inclined floor of the furnace and is met at 6-ft. intervals by other batteries of burners, which insure complete expansion of even the smallest particles. The furnace floor is at an angle of approximately 30° and its length is 22 ft. The ore travels at a rate of 5 ft. per second and is therefore soaked in the heat, averaging 1800° F., for between four and five seconds. It is estimated that 1,000,000 B.t.u. is required for expansion purposes, while our records show a consumption of 2,400,000 B.t.u. per ton through the furnace, indicating a heat loss by this method of 1,400,000 B.t.u. per ton of expanded material.

At the discharge end of the furnace, the expanded material is picked up by a current of air and deposited in hoppers ready for bagging. Particles of foreign matter and partly expanded Zonolite too heavy for the fan to lift with 6-oz. vacuum are dropped and discarded. This suction fan pulls the hot Zonolite through a current of cold air, which cools it instantly and toughens its plates. It insures uniformity of weight in the finished product and completes the separation of foreign matter from the pure Zonolite.

The latest model of the Schundler expanding unit is designed to reduce heat losses to a minimum and a model of this furnace is now operating at the Joliet factory, while a large unit is under construction at the Schundler Long Island City plant. This machine comprises a battery of speed-controlled revolving tubes, indirectly fired with oil. Its operating flexibility will allow the production of all grades and types of exfoliated Zonolite at a much lower cost than the other experimental units described above.

The average weights of commercial expanded Zonolite are as follows:

Mesh	Pounds per Cubic Foot
−½ +¾	4½ to 6½
−¼ +10	5¾ to 7
−10 +28	6¾ to 8½
−28 +pan	10 to 14

These weights are determined by pouring the materials into a box 12 by 12 by 12 in. A 30-lb. weight, 12 by 12 in. in size, is laid on the material without pressure and removed. The box is then refilled to the top to compensate for settlement, again compressed with the weight and refilled until the weight rests even with the top of the measure. The box full is then weighed and the result given as weight per cubic foot.

Settlement tests performed on a vibrating machine operating at the rate of 60 rotary and 60 vertical vibrations per minute, checked after 10,000 vibrations, showed no settlement when the material was first tamped into the machine. A settlement of slightly over 5 per cent was noted in tests in which the Zonolite was loosely poured into the vibrator.

Zonolite is chemically inert. It can be made plastic with any type of binder and it is foolproof in that it requires no skill in its installation. It is uniform and constant in its qualities. Zonolite has fairly good dielectric properties; the voltage necessary to cause a breakdown between two uninsulated joints 1 in. apart (the entire area filled with Zonolite) was reported by the Ontario Hydro-Electric Power Commission as 14,000 r.m.s. volts. J. C. Peebles, of Armour Institute, reports that the porosity of Zonolite results in ionization of the air at a voltage of 20,800 volts.

Properties and Commercial Applications of Zonolite

As a thermal insulating medium, Zonolite is in a unique position. Its field of service ranges from the insulation of dry-ice containers chilled to 60° below zero to protection for the tops of open-hearth furnaces with standing temperatures higher than 2800° F.

The myriad air-cells of expanded Zonolite give it a *K* factor and heat-resistance value equal to or better than most thermal insulating materials. Thermal conductivity (*K*) of the intermediate size (−½ in. + 10 mesh) of expanded Zonolite, expressed in B.t.u. per sq. ft. per in. in thickness per hr. per deg. temp. difference at the mean temperatures listed is as follows: 0.363 at 70° F.; 0.38 at 100°; 0.46 at 300°; 0.56 at 500°; 0.635 at 700°; 0.76 at 900°. Tests were made by the hot-plate method. The samples tested were 24 by 24 by 2 in. and tamped to a density of 7.1 lb. per cubic foot.

In loose form, Zonolite granules are an excellent filling material. They pour readily, they stay where they are placed without undue settlement and are unaffected by heat, cold or moisture. The use of loose Zonolite is increasing rapidly in the field of home insulation. It is replacing rock wool, diatomaceous earth and powdered gypsum board in bake ovens, water heaters, thermal jugs, food containers, safes, stoves, furnaces, refrigerators and other insulated industrial units.

The golden plates of Zonolite remain bright and untarnished and are not subject to oxidation. The coefficient of emissivity of golden Zonolite is comparable to that of metallic reflectors of radiant heat. See Table 1.

TABLE 1.—*Radiation Losses of Zonolite Compared with Other Materials*

Material	Radiation Losses, B.t.u. per Hr. per Sq. Ft. Surface			
	1° F.	100° F.	200° F.	300° F.
Brass	0.027	0.436	1.187	2.347
Zonolite	0.031	0.502	1.365	2.699
Aluminum paint	0.068	1.091	2.968	5.868
Lime mortar	1.160	1.856	5.046	9.976

As a light-weight aggregate in insulating plasters, cements, nailing concrete and other plastic hydraulic materials, Zonolite adds its insulating and sound-deadening properties to its weight-saving advantages.

One part of Zonolite mixed with three parts of ordinary neat gypsum plaster, by weight, produces an insulating plaster weighing 29.1 lb. per cu. ft., with a K factor at 75° F., mean temperature of 0.90 B.t.u. per hr. per sq. ft. per in. thick per deg. temp. dif. By comparison, the standard brown mortar mix consisting of one part of neat gypsum plaster and two and one-half parts of Ottawa sand, by weight has a K factor at the same temperature of 2.45 B.t.u. and a density of 105 lb. per cubic foot.

No tests or comparisons have been made of the acoustical values of the above formulas, although Dr. Paul E. Sabine has made a series of tests on the Zonolite plaster mix. The plaster was applied ⅝ in. thick to a Steeltex partition. It was left under the trowel and exposed to the sound. The absorption coefficients were recorded as shown in Table 2.

Loose Zonolite has been widely used in motion-picture sound studios where it is important to confine noises to a given area. For this purpose, panels are constructed of 2 by 4 uprights faced on one side with fiber board and on the other with fly screen. The area between the studding is filled with loose Zonolite. Dr. Vern O. Knudsen reported absorption coefficients on these panels as given in Table 2.

Johns-Manville Corporation produces a molded acoustical tile of Zonolite and a binder, which can be painted and cleaned and provides splendid absorption values at all cycles. Tests of the Johns-Manville tile were made by the U. S. Bureau of Standards, as given in Table 3.

A series of exhaustive exploratory tests was made at Purdue University, to determine the highest percentage of Zonolite that could be used in concrete mixes without dangerous losses in structural strength. The proper grading of the aggregate, moisture content, methods of drying and molding were also studied. Maximum Zonolite density (9.3 lb. per cu. ft.) was gained through the mixing of 37.5 per cent of fine aggregate (−8 + 100 mesh) with 62.5 per cent of coarse aggregate, grading from ⅜ to 30 mesh. The amount of water that gave the best results was 148 per cent, based on the combined weight of cement and Zonolite. Tests were made on mixes of 1–6, 1–8, 1–10 and 1–12 cement and Zonolite by volume. The 1–6 ratio produced the most satisfactory concrete. Air-stored specimens had about 30 per cent greater strength than water-stored specimens and rod-tamped samples were stronger than those that were merely poured into the molds by hand. These tests proved that Zonolite could be used to good advantage for nailing concrete or light partition blocks but would hardly be suitable for load bearing or heavy masonry uses.

TABLE 2.—*Absorption Coefficients of Zonolite*

ZONOLITE PLASTER MIX ⅝ IN. THICK ON STEELTEX PARTITION

	C-2	C-3	C-4	C-5	C-6	C-7
Cycles	128	256	512	1024	2048	4096
Decibells	0.08	0.11	0.08	0.09	0.08	0.15

REPORT ON LOOSE ZONOLITE BY V. O. KNUDSEN

	C-2	C-3	C-4	C-5	C-6	C-7
Cycles	128	256	512	1024	2048	4096
Decibells	. . .	0.30	0.306	0.294	0.30	0.238

TABLE 3.—*Johns-Manville Tile*

Sample	Thickness, In.	Treatment	C-2 128	C-3 256	C-4 512	C-5 1024	C-6 2048
1	1	One coat of flat paint brushed on	0.05	0.39	0.86	0.73	0.62
2	1	One spray coat and three brush coats	0.11	0.45	0.85	0.64	0.55
3	¾	Unpainted	0.04	0.23	0.83	0.76	0.73
4	¾	One spray coat and three brush coats	0.07	0.33	0.85	0.67	0.52

Safe manufacturers have learned to mix Zonolite with alkaline earths in the construction of insulated safe walls with the greatest heat resistance yet known for this type of insulation.

As the sound-absorbent medium in a widely used silencer or muffler for internal-combustion engines, Zonolite eliminates the necessity for baffle plates and increases the power of the engine, in addition to increasing the sound-absorbing and muffling value of the silencer.

Graded Zonolite granules, blended with mineral fibers and a refractory bond, produced a plastic insulating cement, which has found wide favor in the steel industry. More than 50 steel plants in the United States and Canada have open-hearth furnaces now completely insulated with one or another type of Zonolite insulating cement. These furnaces are showing large savings in fuel costs and increased furnace life.

Experience has shown that a 5-in. coating on furnace roofs; 3 in. on vertical walls above the charging floor, on regenerator roofs and neck arches; and from 1 to 2 in. on regenerator and slag-pocket walls offers the most satisfactory results. Such insulation considerably increases the life of the furnace refractories, since it allows the use of lower velocities at the port. In addition, the soaking of the silica brick in higher mean temperatures causes a conversion in the crystalline form of the brick to a more stable structure. The low temperature difference between the hot and cold sides of the brick compels a uniform expansion throughout the thickness of the brick, removing the hazard of pinching and cracking caused by heating and cooling bricks that have been subjected to localized expansion. Table 4 gives the linear shrinkage of Zonolite cement when subjected to soaking heat for 24 hr. at the temperatures listed, together with loss in weight and volume changes. The modulus of rupture on oven-dried samples of Zonolite insulating cement is 63.3 lb. per sq. in. This cement has already gained favorable recognition and considerable use in the oil, rubber, automobile and railroad industries as well as in the steel mills. It is preferred in these fields for its reclaimability and ease of application in addition to its high insulating values.

TABLE 4.—*Changes in Zonolite Cement under Soaking Heat for 24 Hours*

Temperature, deg. F....	600°	800°	1000°	1200°	1400°	1600°	1800°	2000°
Linear shrinkage........	0.16	0.36	0.43	0.47	0.50	1.18	3.62	7.37

Temperature, deg. F....	662°	874°	1077°	1175°	1475°	1640°	1776°	1883°
Volume shrinkage.......	0.74	1.24	1.37	1.37	1.51	4.68	9.13	14.62
Loss in weight...........	1.86	1.86	4.34	4.34	6.52	8.07	8.15	8.57

Pipe covering, secondary refractory brick, insulating refractories, blocks and molded articles of many types are at this writing in the course of production in a dozen research laboratories.

Illinois Clay Products Co. of Joliet, Ill., is now in commercial production on a light-weight insulating brick which they recommend for use up to 2200° F. when used against refractory walls, and at temperatures below

2000° F. when used independently in walls that are not to bear loads. These bricks weigh about 20 oz. each and may be nailed or cemented into place, as the occasion demands. They may be used freely in furnace construction for both inner and outer walls. The transverse strength of these bricks is reported by the American Refractories Institute Fellowship of Mellon Institute as an average of 27.97 lb. per sq. in. at room temperature. At 1000° F., the average strength is reported at 39.7 lb. per square inch. Table 5 was made by the J. L. Finck Laboratories of Washington, on test specimens cut from fresh brick and tested in a circular, flat hot-plate apparatus, and the tests were run consecutively after a thermal equilibrium had been established between tests.

TABLE 5.—*Conductivity of Illinois Clay Products Light-weight Insulating Brick*

MEAN TEMPERATURE, DEG. F.	THERMAL CONDUCTIVITY,[a] B.T.U.
198.7	0.76
514.9	0.92
878.7	1.05
1,339.3	1.45
1,791.8	2.55

[a] Conductivity values represent the amount of heat that will flow in one hour through a square foot area, when the temperature gradient within the sample is 1° F. per inch thickness.

A shrinkage of 0.082 in. took place during the course of these tests, which was compensated for in the calculations.

The expansion feature of crude Zonolite is used to produce sealing compounds and several types of plastic emulsions.

Flaked Zonolite finds wide use as a decorative medium in plaster, textural finishes, wall paper, etc. Powdered Zonolite can be utilized to replace many fillers where lightness in weight, slip, absorption value or chemical inertness are of advantage.

The list of possibilities for vermiculite seems unlimited. The remarkable and peculiar qualities of this mineral are attested by its widespread commercial adoption in so short a time, which is particularly remarkable because of the highly specialized and highly competitive and technical fields it has penetrated.

Economic Results of the New Technique in Phosphate Recovery

By Charles E. Heinrichs,* New York, N. Y.

(New York Meeting, February, 1933)

Abstract†

There are still ample reserves of phosphate in Florida and Tennessee, but the richest low-cost areas have been exhausted. The miners, by the introduction of more efficient equipment, have succeeded in preventing material increases in production costs. Further mechanical improvements could not be expected to keep up with the factors tending to increase costs. While the conventional phosphate "washers" are capable of recovering practically all of the plus 1.5-mm. phosphate from the matrix, approximately 50 per cent of the phosphate values are in the minus 1.5-mm. sizes. Flotation now affords an economical and efficient means for recovering the fine (−1.5 mm.) phosphate formerly discarded to waste dumps. Through the reclamation of dumps and recovery from current washer wastes, flotation has doubled the reserves of Florida and Tennessee. By doubling recoveries from the matrix, flotation reduces capital investment requirement and substantially reduces production costs. It definitely improves the grade of the total production and provides a flexibility of operation that cannot be obtained by any other known means.

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* Phosphate Recovery Corporation.

† The paper was published in *Mining and Metallurgy*, August, 1933.

Uses of Coal in the Ceramic Industry

By H. E. Nold,* Columbus, Ohio

(Columbus Meeting, October, 1933)

Abstract†

High-volatile coals are most desirable for kiln firing. Low-volatile and even anthracite coals can be used successfully. Most periodic kilns use coal and are hand fired. A few stoker installations have been economically successful. Continuous tunnel kilns reduce fuel consumption 40 to 55 per cent. They are rapidly replacing periodic kilns in the manufacture of the finer wares. They have found great favor in the refractory industry and are successfully used in firing almost all kinds of heavy clay products. Fuel oil has found rather wide application in the Southwest and West. Natural gas is a satisfactory fuel for kiln firing. It is the principal competitor of coal and has largely replaced it for firing the finer wares but cannot compete with it in firing heavy clay products except in regions where cheap gas is available. Figures are given showing the relative thermal efficiency of coal and natural gas as used for firing ceramic ware. The ceramic industry uses from 12,000,000 to 12,500,000 tons of coal per normal year. The principal use of coal is in firing the heavy clay products. This potential market amounts to about 9,250,000 tons per year. Coal producers could increase sales by furnishing expert firing instruction.

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* Professor of Mine Engineering, The Ohio State University.

† The paper, with discussion, appears in *Trans.* A.I.M.E. (1934) **108.**

Basic Trends in Mineral Industries Education

By Edward Steidle,* State College, Pa.

(New York Meeting, February, 1933)

It has been said that "the command of nature has been put into the hands of man before he knows how to command himself," and what we see about us gives particular emphasis to this observation. If this condition be granted, the vaunted boast of school and college, of the institution that had the training of this generation, and of its fathers, and of its grandfathers, the boast that education is a "preparation for life," stands exposed as a hollow claim, or as a ghastly, cynical conception of life, or else education has fallen far short of its intentions.

It has been evident for some time that education was not meeting the fullest expectations of society. In recent years there has been a deal of speculation indulged in by both laity and faculty in regard to American education and American civilization, and some few and sporadic changes have been effected. Today, however, there is no longer time for mere hypothetical considerations; we are faced with certain definite facts and confronted with obvious conditions that will not wait for fine-spun theories. If educators do not put their house in order, someone else will do it for them.

The trends in mineral industries education must be in harmony with this general realignment that is today visible on the educational horizon, and only as these trends are in conformity with this movement can they be called sound. Two questions, then, confront mineral industries educators: (1) What are the trends in mineral industries education, and (2) how do these trends correlate with the major educational movements of the day?

SCOPE OF MINERAL INDUSTRIES EDUCATION

Before launching into these particulars, it might be well to define the scope of mineral industries education and to consider its present position.

Mineral industries education embraces three main types of work:

1. *Earth Sciences and Geography.*—Geology, mineralogy, geography—courses basic to all mineral industries and to the humanities.

2. *Mineral Engineering.*—Mining, petroleum and natural gas—dealing with mineral producing industries. Application of earth sciences and all types of engineering to exploitation of natural mineral resources.

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* Dean, School of Mineral Industries, The Pennsylvania State College.

3. *Mineral Technology.*—Fuel technology, metallurgy, ceramics; dealing with mineral processing and using industries. Application of chemistry and physics and engineering to the preparation, treatment and processing of natural minerals.

In spite of this clearly defined field, however, it has been only in very recent years that this material and these subjects have been collected under a common head. As a matter of fact, the mineral industries subjects, as one of the large, interrelated groups comprising a natural curriculum, are just now in the process of being brought together. As distinct from the educational problems of engineering, agriculture, science, medicine, or any of the older well defined groups, the problem of mineral industries education is, first, to obtain and insure its identity as a distinct unit, which the older groups have already done, and, second, to organize its instruction in accord with modern requirements. Whether this double requirement is a handicap or not is beyond the point; while the mineral industries at present lack the unity of the older groups, they also are free from hampering traditions and precedents.

TRENDS IN MINERAL INDUSTRIES EDUCATION

There is, then, a strong tendency for the mineral industries subjects to unite as a distinct group; this is a logical trend because of the very nature of the work covered by mineral industries education, as well as because of the needs of industry.

The second trend to be noted is that observable in at least one western university and in several eastern colleges; namely, the substitution of technical courses of five and six years for the traditional four-year curricula. While it is true that these courses are attracting few students, there is considerable discussion under way as to the possibility of unifying freshman and sophomore years, and of carrying this idea into the junior year wherever possible. The technical side of the unified freshman and sophomore years would be designed to develop powers of observation as well as ability to make accurate measurements and think in three dimensions. Under this plan, specialization in technical subjects would not start until the junior year, at which point students whose records do not justify further collegiate training could be dropped without great loss of time to themselves. The trend seems to be to limit the technical schools to students of proved ability and merit, somewhat on the same order as the present schools of law and medicine.

Industry always will be dependent on engineering and technology, but will no longer seek college graduates unless their education has been thorough. If, for certain reasons, the four-year curriculum leading to a Bachelor of Science degree is desirable for the time being, the better student should be encouraged to continue his studies for a Master of Science degree, and it is in this work that the greatest amount of specialization should be done. Graduate courses, leading to the doctorate where warranted, should be developed; especially in the earth sciences and mineral technology. Graduate work should have its roots in the research work of the school.

It is possible that the conventional four-year college curriculum may be concentrated into three years, which will cover chiefly such academic courses as may be required to fulfill the needs of higher education in so far as preparing young men and women for citizenship is concerned. If this is to be the case, mineral industries courses might be set up as two years of advanced instruction leading to a master of science degree or to a professional engineering degree, and this may be followed by still more advanced, specialized studies in mineral industries for exceptional students.

The third trend that may be observed in mineral industries education is a return to more fundamental courses in applied science and fewer in the specialized descriptive class. Less stress is being placed on laboratory equipment, except that necessary to demonstrate principles. Many of our mineral industries schools or departments are loaded with white elephants; this has a direct bearing on cost of instruction. Small laboratory units to demonstrate theory is all that is required. Home-made equipment wherever possible will result in better teaching and a more efficient use of available equipment may be made. For example, assay equipment could be used for slag synthesis, heat-treating experiments, clay-burning tests, etc., as well as for assay instruction.

Changes in Curricula

The curriculum in petroleum and natural gas engineering is a particularly good example of these changes. Ten years ago the petroleum industry had dire need of men trained in technique, able to drill oil wells, run viscosimeters and operate crude stills. Today there is a surplus of men ably equipped to perform the routine functions of engineers in an integrated oil company, but who lack fundamentals necessary for leadership.

Being relatively new, curricula in petroleum and natural gas engineering have been passing through a period of experimentation. At the same time, recent developments in the technique of refining, deep-well drilling and production and secondary recovery methods have shown the necessity of more thorough training in fundamental science, particularly in mechanics, thermodynamics, physical chemistry and the elements of chemical engineering. For this reason correspondingly less emphasis is now being placed on those courses whose content is largely composed of material descriptive of oil-field and refinery apparatus and practice. This is a logical development, for fundamental principles endure, whereas industrial practice is in a constant state of flux.

The same thing is happening in all of the curricula; rapidly changing conditions in industry require that the technical graduate should be well grounded in fundamental science so that, as changes are demanded, or as new processes displace older ones, he may keep in step with the development. In geology the curriculum is undergoing such a gradual evolution. At first, a classical foundation was considered essential. Later, engineering and technical subjects were stressed and a period of mass production of mining and petroleum geologists ensued. At present, the emphasis is being placed on thorough training in the sciences, such as chemistry, physics and physical chemistry, in order that the student may continue his training in a graduate school and eventually be well prepared for a research, teaching, or industrial career. As in the medical, legal and other professions, the time is near at hand when a student of geology with only four years training cannot enter his life work with any expectation of success.

Geography has been developing rapidly as a college subject, frequently fostered at first by departments of geology. At no time in history has there been such need for taking stock of natural resources and for critical examination of the ways in which these resources are being used. The economic geographer, particularly when his work is combined with mineral economics, is in a position to render real service of this sort.

In the future, we must work with social and economic facts. Future planning in government and industry will be on the basis of facts, and in discovering and interpreting these facts the geographer, through regional economic geographic studies, will play an important role; likewise, it is necessary that the future program in mineral industries education be guided by studies in mineral economics which may well be carried on by an economic geographer.

The mining engineer in an earlier day was supposed to be trained for all eventualities. He was expected to find and open all mineral deposits, mine and concentrate the ore, leach or smelt the metals, and even refine it for marketable use. He did all these with admirable efficiency and remarkable adaptability, but the increasing magnitude of his job forced him to specialize on the more strictly mining phases of his work. It would seem that the conventional mining engineering curriculum has seen its best days. Geology, mineralogy, mineral economics, beneficiation processes, application of power underground, and mine mechanics are now considered the more important subjects in the training of a mining engineer.

Fuel technology is one of the newer courses of study developed by schools of mineral industries, and must be added to curricula at least in coal-producing states. In foreign universities and technical schools, fuel technology has long been an important course of study. This was a natural consequence of the fact that fuel reserves were not of the quality originally existing in this country. For example, the German brown coals needed special treatment for many processes and fuel technology early reached a high plane of development in Germany. The time has come in this country when similar conditions exist, and fuel technology will be a generally accepted curriculum in our colleges and universities.

Fuel technology is by definition concerned with the systematic knowledge on which fuel-producing and using industries are based. The curriculum should not only give fundamental training in chemistry, physics and allied subjects but should include such courses as origin and constitution of coal, mineralogical and geological structure of the coal measures, pertinent problems relating to exploitation that will provide the desired sizes of product, physics of cleaning and beneficiation, classification of coals for special uses, thermal reactions, coking, complete gasification, hydrogenation, et cetera.

The metallurgist, originally trained largely in mining and chemistry, today is aiding and directing expansion in size of metallurgical application, development of new processes, creation of new alloys, adaptation of new appliances, electrical melting, heat treatment, exact temperature measurement, and automatic control. Added to these is the development of physical chemistry and the gradual realization that the laws developed in that science are fundamental to most of the principles of metallurgy. The microscope and the spectroscope have added much to the metallurgist's responsibility, as well as his capabilities. Physical and mechanical explanation is forthcoming for every behavior of a metal, and mysteries that formerly existed are giving way to logical physical explanations.

Obviously, therefore, the trend in metallurgical education is to devote more time to physics and its application, to light, electrical theory, mechanics, atomistics and mathematics.

In ceramics, where the first instruction was given in 1894 at the direct demand of the industry for more technical information (just as the demand is now arising for fuel technologists), the curricula have been designed largely to satisfy the immediate demands of industry. Because of a lack of knowledge of the science of ceramics, and because until recent years the industries required a ceramic graduate familiar with general plant engineering as well as applied ceramics, our ceramic education has been too general in engineering training, and too specialized in practical ceramics. The present economic situation is teaching the ceramic industries that what they need in a ceramic graduate is not so much a man who will keep their plant operating properly as one who will keep their products of high uniform quality, and who will develop new products and new uses for their products. This requires technical ceramics. The present trend in ceramic education, therefore, is to develop technologists well founded in the fundamental sciences; mineralogy, ceramic petrography, chemistry, physics, physical chemistry, mathematics and mechanics.

Requirements for Teachers

This trend toward fundamental science in each of the curricula, and the teaching of the specialized subjects as the application of a particular science, turns the attention automatically to the teaching staff. Perhaps no statement of the requirements sums up the particulars better than does that of Dr. Edward Orton, Jr., in an address he made at the dedication of the Edward Orton Memorial Library at Ohio State University, Oct. 15, 1920. Dr. Orton said:

> The teacher cannot only teach—he must be taught. He cannot always give out—he must himself take in. He must rejuvenate his jaded faculties with new contacts with the world about him. Research and study are the only avenues to this replenishment of his soul, and hence contact with scientific research and opportunity for study are a sine qua non for good teaching. They must be made not only possible, but necessary to a good teacher's environment.

In facing the new era in mineral industries education more consideration is being given to contents of courses and to methods of teaching. Classroom and laboratory instruction are being scrutinized in detail; "cook-book" methods no longer suffice. There must be less story telling and more searching. It is possible that many instructors are not adequately trained for the work ahead of them. This may mean sabbatical leave for the purpose of doing graduate work. There can be no hope for any instructor, particularly a young man, who wastes long vacation periods, who has to be prodded, and who cannot create his own opportunity. He that has a true teaching and scientific spirit will spend his time in industry, on a research problem, or in creative writing. Results only will tell the story.

Heads of departments may term their headquarters "offices"; other members of the staff would do better to use the term "workshop" or "private laboratory" and, of course, these should be equipped and used accordingly. The administration can assist materially in developing proper attitude on the part of the teaching staff toward research by providing ways and means for its accomplishments, and by removal of hindrances such as excessive teaching burdens and needless committee work. More attention must be given to library facilities in the mineral industries. Such libraries should be in charge of technically trained librarians who know how to make books useful tools and can impart this knowledge to the students.

This new set-up, and the constantly changing conditions, both point naturally to individual and organized research. No matter how small the school, research by the instructional staff is an essential concomitant to rich and vital teaching. The larger mineral industries schools, and especially schools in the land-grant colleges, should have experiment stations leading the way in mineral industries theory. They should not be set up as independent organizations, but should be looked upon as the organized expression of the research spirit of the school. Each member of the teaching staff, as well as the full-time research staff of the station, should contribute, and the heads of departments should be the leaders in their respective fields. Under this plan researchers will not become "solo" workers and will have the entire resources of the institution, mental as well as physical, back of their investigations. The consequent close and intimate contact with research is stimulating and helpful to the undergraduate, to the graduate student, and to the instructional staff. As by-products, the contacts made with industry through the experiment station work of the school serve to keep the two in harmony, and a well established and properly equipped mineral industries experiment station becomes the logical agent for making such state investigations as require laboratory and research facilities.

Providing research facilities for the state does not end the direct service rendered by a school of mineral industries off the campus. In land-grant colleges at least, the trend toward extension service has found the mineral industries division contributing its full share. This service is making the findings in the laboratory available to the man in the plant, and it is also raising the general educational plane of the worker. As the extension service expands in slowly widening circles, it is noticeable that the man in industry is making more and more use of the results of application of theory, and is speaking the language of the laboratory. The college has left its cloister; it is carrying its message into workshop and home throughout the land.

Summary of Trends in Mineral Industries Education

To sum up, the immediate trends in mineral industries education are:

1. Consolidation of underlying subjects under one unifying head.
2. Tendency toward creation of professional schools limiting technical instruction to work of advanced grade.
3. Approach of mineral industries as the application of fundamental science.
4. More research conducted by instructional staff.
5. Establishment of experiment stations as organized expression of research spirit.
6. Dissemination of information through extension service.

DEVELOPMENT IN MINERAL INDUSTRIES EDUCATION IN RELATION TO MAJOR MOVEMENTS IN COLLEGE EDUCATION

Having reviewed the directions in which mineral industries education is developing, we may now ask whether these developments are sound;

whether they coordinate with the major movements of education that are apparent in the making, and with that of college education in particular. This question is of crucial interest; it is of as much, if not more, importance to the ultimate well-being of mineral industries education as is the definition of the trend in this field. For on the answer will depend whether mineral industries education is in a favorable position to develop with the changing educational world or is moving off at a tangent.

To understand what is happening in the colleges—and the wind is full of straws to show that colleges are changing—we may recall that the college has become an integral part of the American system of education, so that a critical examination of education as a whole in this country may reveal the direction in which the college must turn.

Starting at bedrock, then, the first question is: Why, after all, should the public support institutions of learning? and the second, What is the basic philosophy, or the public expectancy, of education in the United States?

Without attempting to go back too far, it may be asserted that the brief for education as a function of the state was filed in America when the Declaration of Independence asserted that all men are created free and equal, and that they possess certain inalienable rights, such as life, liberty and the pursuit of happiness. The corollary logically follows that under a government based on these ideals the state is obligated to insure favorable conditions for their attainment; to assure that all citizens shall have at least an equal start in life; and this leads inescapably to the conclusion that a certain minimum of education is an obligation of the state. As a philosophical conclusion, public education as a function of the state is inextricably bound up with the formation of this nation.

But once established as a nation, a vital and compelling reason for putting this philosophical conclusion into practice developed. Early elections demonstrated that a government dependent on an illiterate and ignorant electorate could not long survive; the very continuance of this nation as a republic was recognized to be dependent upon the elimination of ignorance. Education became the touchstone for citizenship. Thus the whole philosophy behind our educational system has been, first, to insure each child an opportunity to partake of our common intellectual heritage, and, second, to promote the development of better citizens. The minimum for the first objective is achieved when the child is taught to read, but what the minimum for the second objective is has not yet been determined. From the foundations on which our educational system is based, however, this much is evident: beyond teaching a child to read, the expenditure of public monies for education may be advocated as sound governmental economy only as a means to produce better citizens.

Passing by, for the moment, the definition of better citizens, the approach of education to this American desideratum, and the outcome of this attempt, may be rapidly summarized. Starting with the observation that grinding misery and abiding virtue rarely go hand in hand—an observation so universal that we all know exceptions—the particular belief that material dependency and good citizenship are incompatible easily followed. Hence the conviction that sturdy citizenship is dependent on a measure of economic competence became general and, consequently, the desirability of educating young men and women so that they may advance materially has been generally conceded to be in the public interest. Our land-grant colleges are the national expression of this conviction.

Without doubt this immediate, material objective of education has been amply met by the American high school and college; has been overwhelmingly met, in fact, until the conception of material advance as an aid to sturdy citizenship has been completely overshadowed by the immensity of the material success. In recent years we have been educating for bigger and better success; and we got it—and we also got 1929.

This present crisis, which is as much a social and moral debacle as an industrial depression, has been cited as an indictment of current educational practices and procedures. When a hundred years of free public instruction finds America utterly bewildered by this catastrophe, completely unprepared for any change in the tenor of life, there may be some ground for the accusation.

Even so, American belief in the value of education to a democracy has not yet been shaken, but our people are directing a critical search into all of its practices. We are beginning to see that education has been blinded by its material achievements; that it has been beguiled from the straight and narrow path into the primrose bowers of success; that it has fallen short of its own high intentions. The public is asking, and it has a right to ask: Where are the leaders you set out to train? Where is the preparation for life you promised?

All this may seem far removed from mineral industries education, but mineral industries is only a branch of the college, and until we have defined the fundamental reasons for the existence of a college and know wherein it has succeeded or failed, we are in no position to set up a particular curriculum in that college. Without this understanding we are simply molding the old clay, the clay that is found wanting in the emergency today, into new shapes; shapes that will break as easily as the present ones do, when the strain is put on them.

If this analysis is correct, if the college is to hold public confidence and receive public support, it must return to the fundamental purposes of education; it must produce graduates with a wealth of understanding not even approached today; it must turn out young men and women who are competent as citizens as well as competent as technicians, engineers, and bond salesmen.

Obligations of Mineral Industries Education

Primarily, then, as part of a college, mineral industries education must turn out better citizens and, secondly, in claiming to be mineral industries education, it must of necessity produce trained technicians. Unfortunately, as our curricula are set up today, this order has been reversed—and too often the amount of time available in college after all the technical studies have been included allows but the barest touch of other subjects.

Our problem is to reverse this tendency. We must so educate that students will view technical advance in the light of social effect, and, being professional men, observe social requirements as technical problems. How to obtain these ends is a matter of varying opinion, but if we can define our aims distinctly and clearly enough, different methods may be measured by a uniform standard to the benefit of all.

It is possible, however, to go a step farther here, to sound a warning against the first reaction of educators in technical subjects to the acceptance of these conclusions: namely, the assumption that to balance technical curricula, and to point toward better citizens, all that is necessary is to include more subjects in liberal arts. For it may be asserted that the graduates from the stratified and ossified liberal arts curricula have failed as miserably, in understanding and as leaders, in this national crisis as have the professional and technical men. Yet it is safe to say that in the humanities, in history and economics—generally in the broad field of human relationships, human endeavors, human accomplishments, and human records—in the basic stuff that comprises the raw material for the liberal arts, we will find that which we need to produce better citizens.

In this region of human knowledge lies the material we need to "educate so that professional students may view technical advances in light of social effect, and to observe social trends in terms of technical requirements," but we cannot accomplish this result by following the present liberal arts methods. Granting that separation of the various activities of man into individual units are necessary for research, the presentation of the findings to students as so many isolated subjects is not productive of understanding and wisdom. To give an undergraduate a nice little package of history, another of economics, a third of political science, each wrapped all by itself, and not one revealing the underlying beliefs and dreams for which man has striven, and suffered, and died, without a single attempt to uncover the psychological factors and reactions that enabled him to do it, is to turn out students who have a jumbled assortment of isolated facts, facts that are relatively meaningless by themselves and are soon forgotten.

This liberal arts material, which scholars have been at great pains to sort out of the unwieldy mass of all human activity for the purpose of study, must be re-integrated, must be fused, welded together again by the instructor if it is to be of value to the student as a future citizen.

Only when we have done this for our students have we given them the fundamentals from which they may develop a sane and balanced perspective; an objective point of view from which they may evaluate the present and have some hopes of foreseeing the future. Only from such an education may we expect our technical graduates to view technical changes in their social results; and only as we succeed in doing this are we consciously and actively striving to produce better citizens. As a personal criterion of our faith in such a program for the college of today, a program that centers once again on citizenship, we, as educators in technical fields, must hold failure in subjects directly bearing on citizenship as a more serious deficiency in a technical student than failure in one of his professional subjects.

Better Citizenship

Now, after having been put off as long as possible, it becomes necessary to define "better citizenship," and to point out what education can do to forward it. The citizenship which this country, as a republic, requires is one that maintains a position of detachment and does not have its entire vision filled with immediate circumstances; one that sees the present as the product of both the past and of the future and that is not deceived by juxtaposition of events into *post hoc* conclusions; a citizenship, in short, that sees and evaluates today as yesterday's tomorrow, and as tomorrow's yesterday. In addition, the desirable citizen knows himself to be but the biological link between his forefathers and his grandchildren; realizes that he has inherited all the world up to now, and that his mite, either for better or worse, will be added to the heritage of his children. From his knowledge of the history of man, of the rise and fall of great families, of principalities, of nations, he knows there is little he can do directly even for his children, and that no effort of his can assure that his great grandchildren will have even bread. He is forced to the conclusion that, as an individual among the millions on this earth, he is unimportant, but that his actions are of the utmost importance, and that all he can do for his family, for his country, for all of mankind, lies in his contribution to the general heritage, and that the contribution of the humblest worker is as valuable as that of the mightiest financier. He also knows that, for good or ill, whether he wants to or not, his every action or inaction contributes to this general fund.

That which education can do to promote this type of citizenship is to build up the perspective; open the long vista of the past in such a manner that it becomes vital today; link cause to effect through the ages;

delineate the ends toward which the efforts of man have been directed, the ideals which motivated him, and point out the results, trace their influence through the years and show how it affects the present.

This is the education on which it would seem to be sound economy to expend public funds; it is the direction in which education, and particularly college education, must move if it is to fulfill its high promise. However awkward or inept this appraisal may be, it marks out a major movement with which the trends in mineral industry education may be compared; it gives a measure from which it may be stated whether or not these trends are sound and in the public interest.

If we can agree with the definition of better citizenship to be sought by education as advocated here, and can accept the obvious responsibilities of the college which follow, then we may conclude that the major trends in mineral industries education are in harmony with the major educational movement, and are sound.

Mineral Industries Education Must Progress

The first two trends in mineral industries education noted, which are generally the consolidation of technical instruction into advanced work, meet the requirements indicated in the conclusion that only students who show an aptitude to become better citizens are worthy of the trust implied in technical training.

The third trend in mineral industries education, that of approaching mineral industries instruction as the application of fundamental sciences, which implies the trends toward more faculty research and experiment stations, seems to fulfill the requirement that we provide a thorough professional training. The final trend noted, that of extension service, meets the obligation of the college to serve as a source of enlightenment for the whole community—a consideration of a function of the college that is beyond the immediate scope of this paper.

It seems reasonable to assume that mineral industries educators have done a fair job in teaching young men how to earn a living and are pointing now in the proper direction. The problem ahead of educators, however, is by no means an easy one: it necessitates an improvement in technical education and requires a realignment of cultural and humanistic subjects. In this new program the teacher will probably become an even more important consideration than he is today, and it behooves us to start selecting and training our younger faculty members now for this new role. In accomplishing these ends, it is evident that mineral industries educators must unite into a section on education such as is inaugurated in the Institute. Probably there are more than 1000 men engaged in mineral industries education in the United States, including the earth sciences, but they have never united with a view to the promotion of progress in their chosen field. Mineral industries education will never come completely into its own, or give maximum service, until the men engaged in the work are thoroughly organized and have a progressive program.

It is imperative that from now on mineral industries educators stand on their own responsibility and solve their own problems; that they do not permit their work to be overshadowed or minimized by the work in another field. Not only should the results of a comprehensive investigation of mineral industries education aid in meeting the present economic program, but their application should put schools and departments of mineral industries in the strongest position for sound progress.

DISCUSSION

(*Charles H. Fulton presiding*)

J. A. Carpenter,* Reno, Nev. (written discussion).—The easiest discussion to prepare on a friend's paper is to compliment the author and agree with him; but Dean Steidle, knowing this, has requested, "criticism from all angles" in order to test out the soundness of his well presented ideas. As I differ with many of these ideas, I will criticize frankly, and in turn, invite similar treatment of my opposing ideas.

The author, in opening, paints a dark picture of the alleged failure of college education, presumably including that in mineral industries. If you seek the specific charges you do not find them; however, chills run up our spines for the safety of our positions as college professors, when we read that if we do not put our mineral education house in order, someone else will.

Why! Only in the nineteen twenties, we were swelling with pride over the fine mining courses we were giving the boys and over the avidity with which the mineral industry employed and praised them. Now what can the matter be? Have we caused this depression by overeducating our students to the advantages of the invention and use of labor-saving devices, or have we failed to add sufficient cultural courses and economic courses to have made them the chosen intellectual and moral leaders that were to steer us clear of the present economic morass? Alas! even our most outstanding mining engineer was rejected by the people for one of the legal and political profession! Surely we should be in sackcloth and ashes and humbly confess our sins and seek out our omissions!

We read on with bated breath to hear the charges the author is going to present against us. First, he says that the mineral industries education has not yet become a distinct educational unit, as, for instance, agricultural education, and "has not yet organized its instruction in accord with modern requirements." However, he suggests this may not be a handicap, which in turn suggests that it may have been the other educators that put us in the present economic pickle.

He speaks with favor of the six-year course of certain mining schools today as a possible good plan for all, with elimination of more students after two years and more thoroughly educated graduates, all on the presumption that the mineral industry demands it. This last point I consider a very debatable one. If this uniform six-year course cannot be attained at once, then he recommends that the best students be encouraged to take Master's degrees or, as an alternate, a three-year general college course followed by two years of mineral industries courses, or five years in all. He points out that the petroleum engineer, the fuel, ceramic and underground engineer, and the metallurgist, all have need of specialization, evidently, that cannot be given in four years of common work. Yet the author states that the present trend is a return to more fundamental courses in applied science with fewer specialized descriptive courses, indicating less need for diversification. He stresses less expensive laboratory equipment. However, the suggested substitution of handmade laboratory equipment for undergraduate work would contrast sorely with the most exacting of equipment and measuring devices for Master's and Doctor's work. He advocates increased research work for mining faculties with increased time for it through less committee work and shorter teaching hours.

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* Professor of Mining, Mackay School of Mines, University of Nevada.

As a whole, then, the author believes the path of progress lies in longer specialized courses, less time given to descriptive or practical courses with much greater emphasis on graduate research work for both student and instructor. Evidently the general four-year mining course has been a grave error, and, to quote the author, "it would seem that the conventional mining engineering curriculum has seen its best days."

With these general conclusions I differ widely. I favor the four-year general course of mineral education as given in most schools today, allowing in the senior year a certain amount of electives for moderate specialization, for the following reasons:

First, the four-year course is an excellent balance between two years of scientific preparatory courses and two years of broad mineral industry courses that are or should be based upon the first two years of work. Second, even with expanding ideas on higher education, it represents the maximum time and financial expense that either the average student or state can see fit to expend, especially in consideration of the salaries paid to the graduates. Third, compared with an extended six-year course the four-year course results in the greatest number of well grounded graduates, which should be the ultimate aim in a republican government and especially in all state-supported schools.

The six-year courses for doctors and lawyers with a diploma necessary to practice has resulted in a high-class labor union that the mining engineer howls at when he contributes. However, in envy, many engineers now advocate longer courses and the licensing of only graduate engineers.

In contrast, many of the medical profession and the majority of the general public are advocating more general practitioners and family doctors with less specialization and less cost to the public.

The heads of the mining schools in their zeal for more specialization often forget the homely fact that to support a few specialists in a large organization, at the peak of the pyramid, there must be a host of subordinates in the operating, technical and executive departments well equipped for their work, who are the backbone of the industry. The average mining school had best center its attention upon the education of this host of engineers rather than upon the specialist.

The mining educator tickles his vanity with the feathery idea that his four short years of contact with the students is the epoch-making period in the student's life and that those he considers his best students will be the peak of the engineering pyramid. He only longs for more time to train them!

How surprising, then, to note the number of engineers honored by schools under the simple title of "former students," or famous graduates honored without any reference to their low scholastic work during college years! Even mining faculty members wear pins of high scholarship fraternities granted to them after they became faculty members.

Therefore, true to democratic principles, emphasis in mining schools, especially state schools, should be devoted primarily to the humble undergraduate student, no matter how unpromising he may seem to be, to see that he obtains a good groundwork in the four years on which to build in future years, to see that he receives high-class instruction in the classroom, and lastly, to see that his instructors are not so wrapped up in research, consulting or extension work, that they talk over his head or fail to give him personal attention and inspiration.

The manner in which the graduate in the succeeding years handles his opportunity for study and the practical work before him is a larger factor in his success or failure than his four years of intensive study in college. Therefore, it is as much the duty of the college professors to instil in him a love of learning and enthusiasm as to ground him in the fundamentals of the mineral industries education.

The bright student with a distinctly scientific trend of mind will gravitate naturally into graduate work along a chosen field of the mineral industry. The average graduate while employed in the industry may become so interested in some particular field that he will return for graduate work. Those mining schools that can amply finance undergraduate work with excellent teaching staffs and facilities, and then have excess funds, could well undertake a graduate school, especially in that branch of the mineral industries education for which they are best fitted by staff and location. Thus, a graduate would look to the schools of great oil-producing states for advance work in petroleum engineering, or to those situated in the great financial centers for mineral economics. The number of these schools should be limited, and even then the undergraduate should have first consideration with the faculty.

The author limits his criticism of present-day teaching to criticism of "cook-book" and "story-telling" methods, and asks for "more searching after truths." This theme is worthy of much greater amplification, for the very direct criticism is made by the education departments of universities that the teaching staffs of engineering colleges are poorly grounded in the very fundamentals of the art of teaching. We engineers who have watched the progress of our children through the grammar schools and high schools of today marvel at the improvement over the methods of our day. To teach in a high school of today, the teacher must have spent many hours in college learning the great art of teaching and do much practice teaching under supervision. To teach in an engineering school the requirements are based on degrees held or research and field reputation. The teacher is then usually left alone in the glory of his classroom, never to realize his teaching failures.

The Society for the Promotion of Engineering Education has given much time to the discussion and improvement of this weakness of engineering teaching, although the Wickenden report touched upon this topic very gingerly. The American Association of University Professors is engaged in a comprehensive study of college and university teaching and of methods of improvement, with most interesting reports on this study. Its recent (December, 1932) bulletin is devoted to this subject and is worthy of the most careful reading by all engineering teachers.

A recent book, "College and University Administration," written by two doctors of philosophy, in discussing instructional administration, deplores the lack of training for teaching in college faculties or the proper supervision of teaching, and also deplores the appointment of faculty members based upon proved ability to absorb knowledge, rather than of imparting it. In our own college experience we can recall with pleasure those few professors who both taught well and inspired. This truly is a wide field for improvement, the discussion of which should not be neglected for that of graduate and research work.

The author then devotes much of his paper to the importance of producing graduates with "a wealth of understanding," and "to sturdy and better citizenship." This is a tremendous responsibility to add to the technical professors, especially as the author admits the failure of the very departments in colleges committed to this task! He admits their unrelated courses in history, economics and political science and "their failure to reveal to the student the underlying beliefs, the psychological factors and reactions."

Truly, then, these college departments must set their houses in order or we engineers will teach these subjects ourselves! It is strange why these departments and the education departments do not pick the beams out of their own eyes before they criticize the jumbled courses and poor teaching of the engineers! Yet in our own midst, Dean Bush, of the Massachusetts Institute of Technology, says, "we have held to an antiquated system of a multitude of loosely connected subjects."

It is my belief, however, that these departments serve their purpose well, and that we can leave the teaching of citizenship and responsibility, and love for our brother man, to the impressionable years in the grades and high schools, and especially to those years in the home when the child is taught by the lovely example that his parents set him in living a Christlike life. I humbly suggest that in this particular we might find our greatest present failure.

Our present failure as teachers in the mineral industries, along the line Dr. Steidle suggests, is not in what we are not teaching in our college courses, but rather in the example we set to the students in our daily life on the campus and in our homes, of our relations to our fellow men. The college president in choosing faculty members might well place in the scales, opposite the college degrees and research abilities, the overbalancing items of ability to teach and inspire both in the classroom and in private life. True, to fill a faculty with those possessing all of these desirable qualifications is, alas, a dream of the millenium!

If we must improve our work to save ourselves, before we soar to six-year courses with more specialization and too much graduate and research work, we had first of all best center our attention on the humble four-year student by striving to improve our teaching and the example we set for him.

Broadening Undergraduate Curricula in Mining and Metallurgical Engineering

By C. L. Dake,* Rolla, Mo.

(New York Meeting, February, 1934)

Abstract†

Many proposals have recently been made regarding the broadening of engineering education, often with little consideration of the problems their application involves. The chief purpose of this paper is to point out how many and how complex are the questions that absolutely must be answered, in any conscientious effort to solve the problem of liberalizing engineering curricula.

Among proposed additions, more training than is now given has been urged in public speaking, report writing, business law and contracts, general finance, money and banking, scientific management, general economics, economic and financial history, social problems, labor problems, public ownership, psychology and similar related subjects. The urge in this direction is widespread and persistent. In discussing this problem, the following questions must be given careful consideration, and some absolutely definite conclusion reached:

1. *The Present Curriculum Load.*—Is this load actually too heavy? In the face of this load, is there room to add broadening subjects to our curricula? Can elimination of present subject matter make room for such additions? Can an elective system accomplish the desired end?

2. *Problem of Early Specialization.*—Is early specialization a weakness? Is industry forcing early specialization? Do our varied degrees demand early specialization? Can broadening be harmonized with the present degree of specialization?

3. *Graduate Work to Solve the Broadening Problem.*—Can we add the desired work in the graduate school? Are the proposed additions of graduate caliber? Is it desired to reduce graduate standards? Can present undergraduate work be removed to the graduate school, to make room for desired broadening courses? Is graduate work normally broadening, or is it specialization?

4. *Importance of Proposed Additions.*—Will they increase entrance salaries? Will they increase the engineer's efficiency? Will they raise

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* Professor of Geology, Missouri School of Mines and Metallurgy.

† The paper was published in *Mining and Metallurgy*, April, 1934.

the standing of the engineering profession? Will they make the engineer a better citizen?

5. *Lengthened Course of Study.*—Is this a suitable time to lengthen engineering courses? What may be accomplished by lengthening the course of study? What are the methods of lengthening the course of study? Which of several proposed plans is desirable? How will this lengthening affect enrollment, and would the results be good or bad?

6. *What Is the Relation of This Problem to Present Public School Standards?*

Development of Technical Education for the Petroleum Industry

By H. C. George,* Pittsburgh, Pa.

(New York Meeting, February, 1934)

Abstract†

WITH a growth of 718 per cent in the petroleum industry from 1901 to 1931, there was a corresponding increase in the demand for technically trained men in the industry. This was first met by increasing numbers of college graduates from the old established courses in chemistry, geology and engineering. However, a need began to be felt for scientists and engineers specially trained for the petroleum industry. In 1912 the University of Pittsburgh established the first department in any university in the world that offered major work in petroleum engineering and petroleum geology.

In Europe, the first major work in petroleum engineering was offered in England in 1913 at the University of Birmingham.

The University of California and Stanford University offered major work in petroleum engineering in 1915. Since that time at least 10 other American universities and colleges have offered major work and at least 12 others are now offering some courses dealing with petroleum and the petroleum industry.

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* Head, Department of Oil and Gas Production, University of Pittsburgh.

† The paper was published in *Mining and Metallurgy*, June, 1934.

Applied Psychology in the Payment of Bonuses, Whether for Increased Service or Mine Safety

By Eugene McAuliffe,* Omaha, Neb.

(New York Meeting, February, 1934)

Abstract†

THE management of The Union Pacific Coal Co., which operates a group of coal mines in Southern Wyoming, believes in the value of applied psychology as a productive factor in the work of getting out coal. Operating almost wholly through the medium of mechanical loading machinery, the company has for some years paid a bonus, equivalent to one-half the labor saving made above a determined average output per shift worked, on a given machine. No penalty is attached where the established output is not obtained, but in 1933 the men on one class of machines earned an average premium of $0.633 per shift on 56.4 per cent of the tonnage loaded. On another class of machine, the men earned an average premium of $0.686 per shift on 28.4 per cent of the tonnage loaded. An increase of 36 per cent in tons loaded per man-shift has resulted since the program was instituted.

This coal company has also been very successful in reducing mine accidents through prizes awarded in an unusual way. At the close of the year, every man whose particular mine section passes through the year without a lost-time accident is given a ticket entitling him to participate in a safety drawing where an automobile is awarded as first prize. There are also a number of cash safety prizes which are similarly awarded. Some 1500 persons attend the drawing, which excites much interest and attention.

The particular plan employed, the payment of production bonus awards and safety prizes, is foundationed upon collective rather than individual effort, and, once gotten under way, has proved to be substantially self-enforcing.

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* President, Union Pacific Coal Co.

† The paper was printed in *Mining and Metallurgy*, August, 1934.

Are Too Many Students Taking Mining Courses?

By William B. Plank,* Easton, Pa.

(New York Meeting, February, 1934)

Abstract†

THOSE interested in training engineers for the mineral industry should consider how their men may fit into the general industrial recovery that now seems well started. One hears occasionally that too many engineers have been produced, especially mining engineers, and that the opportunities for their employment are decreasing. Just what is the situation?

Enrollment in American engineering schools has fallen off sharply during the past few years. In 1930–31, the year of peak enrollment in all engineering schools, there were 78,685 engineering students in the United States, but in 1932–33 only 67,584, a decrease of 14 per cent. President Walters, of the University of Cincinnati, in his survey of 74 engineering schools, published in *School and Society* of Dec. 16, 1933, estimates that the engineering enrollment for 1933–34 is 10 per cent under that of 1932–33, or about 60,000. The situation in the mineral technology schools somewhat parallels this general trend, although not to the same degree. Last year there were 4443 students in mineral technology schools, compared with 4687 in 1930–31, a decrease of 5 per cent. This year, with 3995 students, the enrollment shows a decrease of 10 per cent from last year, or a decrease of 15 per cent from 1930–31.

Enrollment has no doubt been influenced by the condition of the mineral industry, just as the enrollment of other engineers has been affected by general industrial conditions. But the mineral industry, which is the first to recover, has not been so seriously affected as have some of the manufacturing industries during the past four years.

This is especially true of coal and gold mining, which have been increasing rapidly in their production during the past year and have given employment to more men. Ten years ago, approximately 1,000,000 men were employed in the coal and metal mines and quarries, compared with probably half that number now. The number of mining engineering graduates in 1934 will be 234 and in metallurgy 252. In other words, fewer than 500 graduates are entering these industries the present year. With conditions improving in the mineral industry, coupled with a falling enrollment in our mining schools, there certainly exists an opportunity for young men now beginning their training for this industry.

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* Head, Department of Mining Engineering and Metallurgy, Lafayette College.

† The paper was published in *Mining and Metallurgy*, May, 1934.

Nationality of Commercial Control of World Minerals

By William P. Rawles,* Washington, D. C.

Abstract†

In the foreword by C. K. Leith, Chairman of The Mineral Inquiry, it is pointed out that the ordinary mineral production tables do not show the commercial control, and that some knowledge of the situation is desirable as a background for the consideration of problems involving mineral raw materials. The report consists principally of tables, accompanied by a brief text, giving the production and an analysis of the control, according to nationality, of 18 minerals: aluminum, chromite, copper, iron and steel, lead, manganese ore, mercury, molybdenum, nickel, nitrates, petroleum, potash, silver, sulfur, tin, tungsten, vanadium, and zinc. For the most part, the production figures for 1929 and the latest figures for reserves have been used. When feasible, the control has been shown at the various stages of production; mining, smelting and refining. American capital was dominant in the production of 10 mineral commodities, including six of the most important; British capital, in four instances; while Spanish, German, Chilean or Chinese interests each led in the control of one. British, French, German, and Dutch capital occupied second place in the control of most of the important minerals. Between two-thirds and three-fourths of the world's mineral production is controlled by British and American capital.

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* Secretary, The Mineral Inquiry.

† The paper was printed as A.I.M.E. *Contribution* No. 41.

Manganese for National Defense

REPORT OF SUBCOMMITTEE ON MANGANESE OF A.I.M.E. COMMITTEE ON INDUSTRIAL PREPAREDNESS

BY C. MINOT WELD,* *Chairman*

ABSTRACT†

THE Subcommittee on Manganese, of the A.I.M.E. Committee on Industrial Preparedness, prepared under date of Jan. 24, 1924, for and at the request of the War Department, a confidential report on manganese as an element in the problem of national defense. This report was later released and published, in slightly modified form, as a separate pamphlet. After the lapse of eight years, in view of the vital position of manganese in the problem of industrial preparedness against a war emergency, the War Department made a request to have the conclusions of 1924 reviewed and brought up to date. To comply with this request the manganese subcommittee was reconvened and exhaustive studies were once more made of the entire subject of manganese. A new report, with specific recommendations, was submitted to the War Department in February, 1932. This second report, with certain specific conclusions and recommendations expressed to the War Department omitted, has now been released and is hereby published.

Following a rather brief discussion of uses of manganese and classification of manganese ores, requirements and sources are dealt with at some length. Domestic ferro-grade reserves are estimated on the basis of price and are shown by means of a curve where tons of metallic manganese are plotted cumulatively against prices ranging from 50¢ to $1.00 per unit; thereby indicating at a glance the tonnage which the Committee believes would become commercially available at any particular price per unit. Foreign sources of ore are described and foreign prices which the domestic producer must meet are considered. The question of a war-emergency reserve is then analyzed, requirements are balanced against apparent resources, and the principal conclusion that emerges is that a war reserve or stockpile is still vital to national defense.

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* Consulting Engineer, New York, N. Y.

† The paper was published by the Seeley W. Mudd Memorial Fund, as A.I.M.E. *Contribution* No. 48.

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